INFERRED WEAK ROCK MASS CLASSIFICATION FOR STOPE DESIGN

A Thesis Submitted to the College of

Graduate Studies and Research

In Partial Fulfillment of the Requirements

For the Degree of Master of Science

In the Department of Civil and Geological Engineering

University of Saskatchewan

Saskatoon

By

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Abstract

Empirical design methods are commonly used for rock mechanics evaluations. An appropriate method of rock mass classification is required to use these empirical methods. There are limitations for rock mass classification methods when access to the ore zone is restricted.

The Cameco Corporation Eagle Point Mine in northern Saskatchewan, Canada, uses the longhole open stope mining method for the recovery of uranium ore. The Modified Dilution graph is used for the prediction of stope hanging wall dilution. The mine currently uses a rock mass classification based on an estimate of the alteration and strength of a rock mass from geological drift mapping. Since this method is highly subjective, point load testing of diamond drill hole core was completed to attempt to correlate the alteration and strength of different rock types to remove the user subjectivity. The results of the testing indicated a general trend of decreasing rock strength with increasing alteration, albeit with considerable scatter.

A repeatable, standardized method of evaluating the stope geometry and inferred rock mass classification for reconciliation purposes was developed. The standardized stope evaluation method removes significant subjectivity currently involved in estimates of stope geometries and the magnitude of dilution. A new lithology based method for interpreting the mine specific geological alteration and strength classification system was developed based on several sources of rock mass classification observations. This resulted in a correlation linking individual rock mass property descriptions between different classification systems for an improved estimate of the Q' classification value. This improved method of estimating the rock classification Q' value, as well as conventional techniques for linking classification systems, was used in a stope reconciliation process to predict open stope dilution.

Twenty-seven stope reconciliation case histories were documented and used to compare predicted and measured dilution, based on three different approaches for estimating rock mass classification values. The results showed a minor improvement in dilution prediction using the approach developed in this study. The systematic stope reconciliation and rock mass

classification approach did highlight areas in the weak pegmatoidal rocks where improved rock classification estimates should be investigated.

Acknowledgements

The initial support for this research project by Cameco Corporation and the Natural Sciences and Engineering Research Council of Canada is greatly appreciated. Special thanks to Cameco Corporation employees both former and current: Richard Basnett, Cory Belyk, Guy Dishaw, and Rob Pagnin (most of all) from the Geology Department, and Jan Romanowski, Alex Pop, and Cristian Gherghel of the Mine Engineering Department.

I would like to thank my committee members Chris Hawkes and Grant Ferguson for providing their feedback on a tight timeline. Many thanks to Donna Beneteau for proof-reading, edits, suggestions, and cheerleading to cross the finish line.

I can't thank Doug Milne enough for showing great patience and guidance through the extended duration of the thesis.

Special thanks to my parents, Gerald and Georgina Greenlay, for showing me the importance of advanced education, but also how to have fun while doing it.

A final acknowledgement to my husband, Glen, for his patience, his understanding, his one good ear to listen, his shoulder to cry on as needed, and for giving me our beautiful daughter, Elisabeth Andrea. She was the true inspiration for completion of this project.

This thesis is dedicated to Donald James Campbell Duthie, who would have been tickled pink to see the final product.

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Chapter 1 - Introduction

Mine design in rock is conducted in a very wide and highly variable range of material properties. Methods for estimating the material properties for a rock mass rely upon a method of estimating several rock mass properties. These estimated rock mass properties are then combined in a single term using a technique called rock mass classification. Rock mass classification is an integral component of engineering projects, including mine design. To be used effectively, appropriate values must be assigned to describe the rock mass. These values can be combined to give a single classification number from which an engineering design can be developed. Using appropriate input parameters is critical to having a rock mass classification value that properly reflects the properties of the rock mass.

This thesis will illustrate the methods of rock mass classification used specifically at the Cameco Corporation Eagle Point Mine, which is a uranium extraction operation. Typically, rock mass characterization is performed on exposed areas of the rock in the vicinity of the extraction zone. Due to the radioactive nature of uranium ore, it is seldom possible to directly measure the rock mass properties for every metre of advancement. Each advance of development in the ore zone is shotcreted, as soon as possible, for gamma radiation shielding of personnel and for primary ground support.

1.1 Background on Field Site Data

The Eagle Point Mine is located in northern Saskatchewan at the Cameco Corporation Rabbit Lake mine site (Figure 1-1). The Eagle Point underground mine excavates uranium ore for use in the nuclear fuel cycle. The uranium ore was hydrothermally deposited in fractures of the Athabasca Basin metasediments. The hydrothermal alteration affected both the fracture surfaces of the rock mass, as well as the fabric of the rock mass itself.

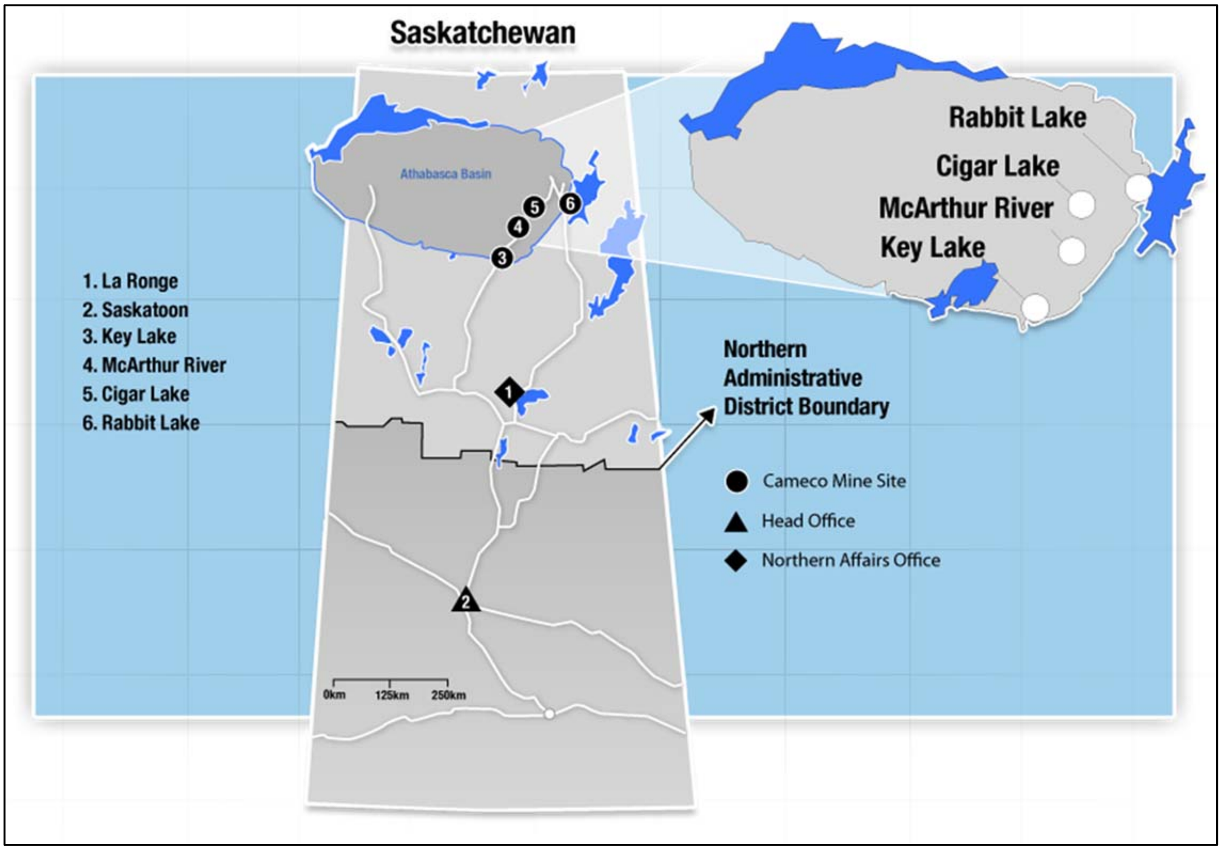


Figure 1-1 Location of Rabbit Lake Mine site (#6) (Courtesy of Cameco Corporation, 2010).

1.2 General Mine Information

A common bulk extraction method used in underground mines is the open stope mining method, and this is the method used at the Eagle Point Mine. With this technique, the ore zone is accessed through overcut and undercut drifts, which are driven along the ore-bearing fault and fracture zones (Figure 1-2). Drill holes are used to blast the rock between the overcut and undercut after which mucking is conducted with remote scoops from the undercut. The support used, height between the overcut and undercut drifts, and the length of stope to be blasted are designed based upon the rock mass properties of the area surrounding the drifts.

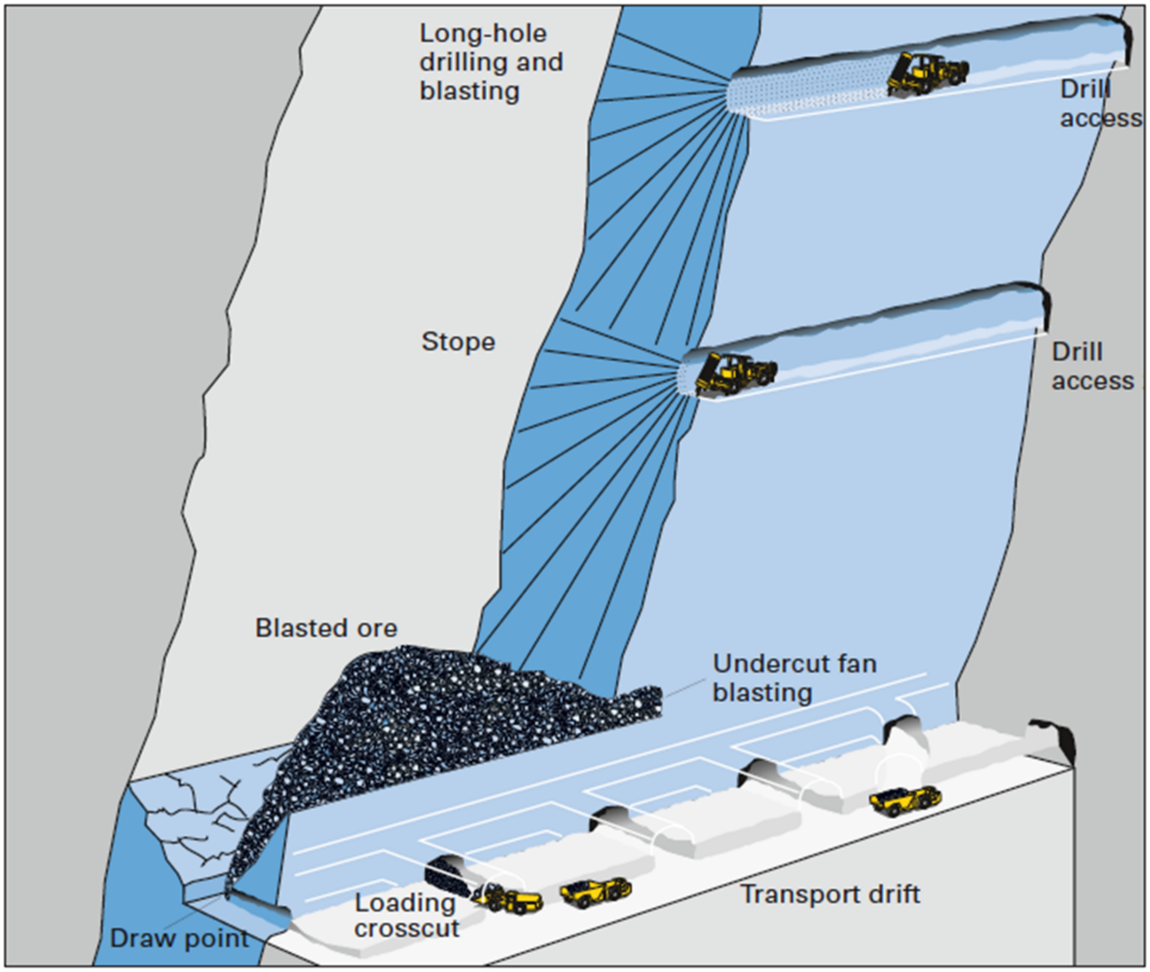


Figure 1-2 Longhole open stoping mining method (Atlas Copco Rock Drills AB, 2007).

Occasionally, waste material is blasted or fails and is then removed either intentionally or unintentionally along with the ore material. Dilution, the amount of waste material included while excavating an ore block (Figure 1-3), can have a major impact on the success of an underground mining operation.   Figure 1-3 shows the geological zone, which is the mineral zone to be extracted, the mining zone, which is the optimum mining shape for ore recovery, and the extraction zone, which is the amount of material that was removed by mining. Planned dilution may be necessary in order to remove all of the ore material. However, unplanned dilution may occur for a variety of reasons such as failure from the stope walls, commonly called overbreak.

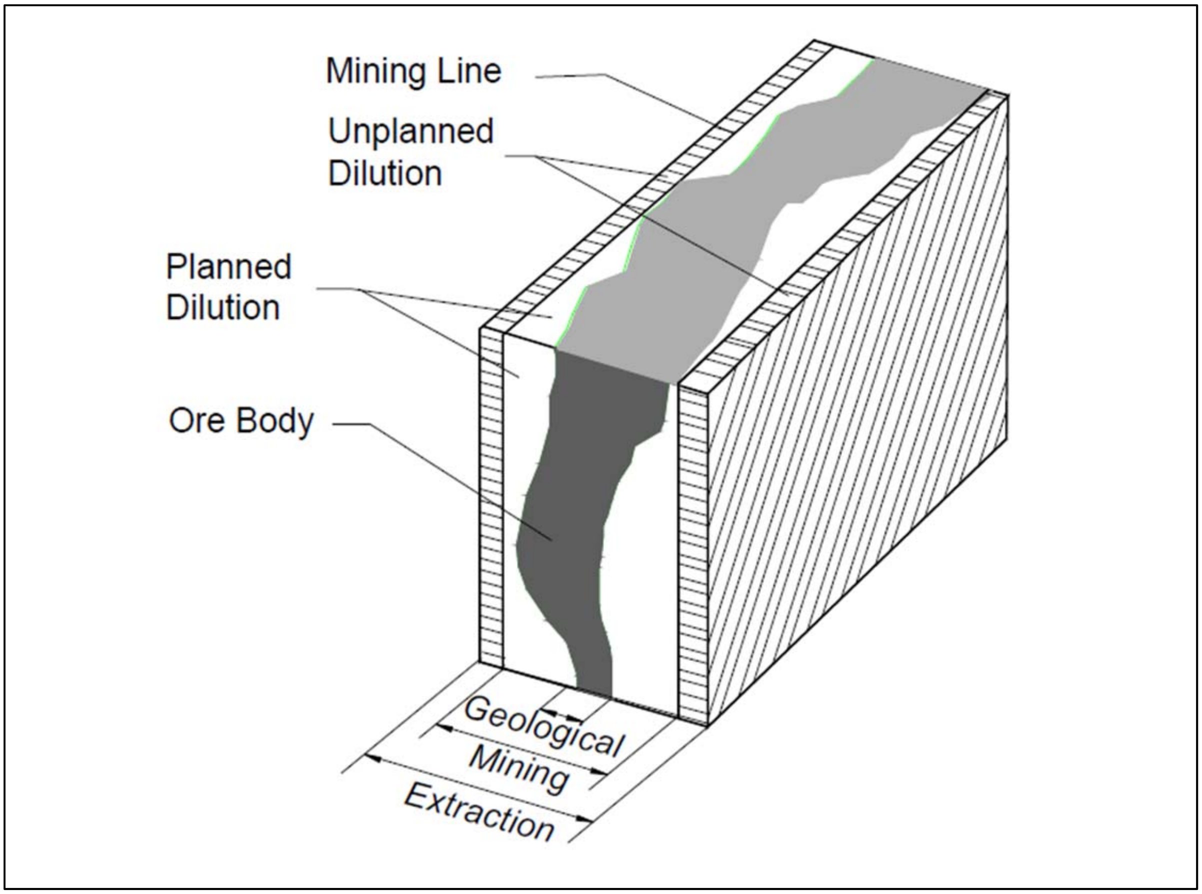


Figure 1-3 Definition of dilution (After Scoble and Moss, From Wang, 2004).

Figure 1-4 illustrates some of the common terms used for the open stope mining method. The stope block is the material to be removed, the overcut drift is the tunnel opening over the stope block, and the undercut drift is the tunnel opening under the stope block. The hanging wall is the upper or overhanging wall of the stope block, and the footwall is the lower underlying wall of the stope block.

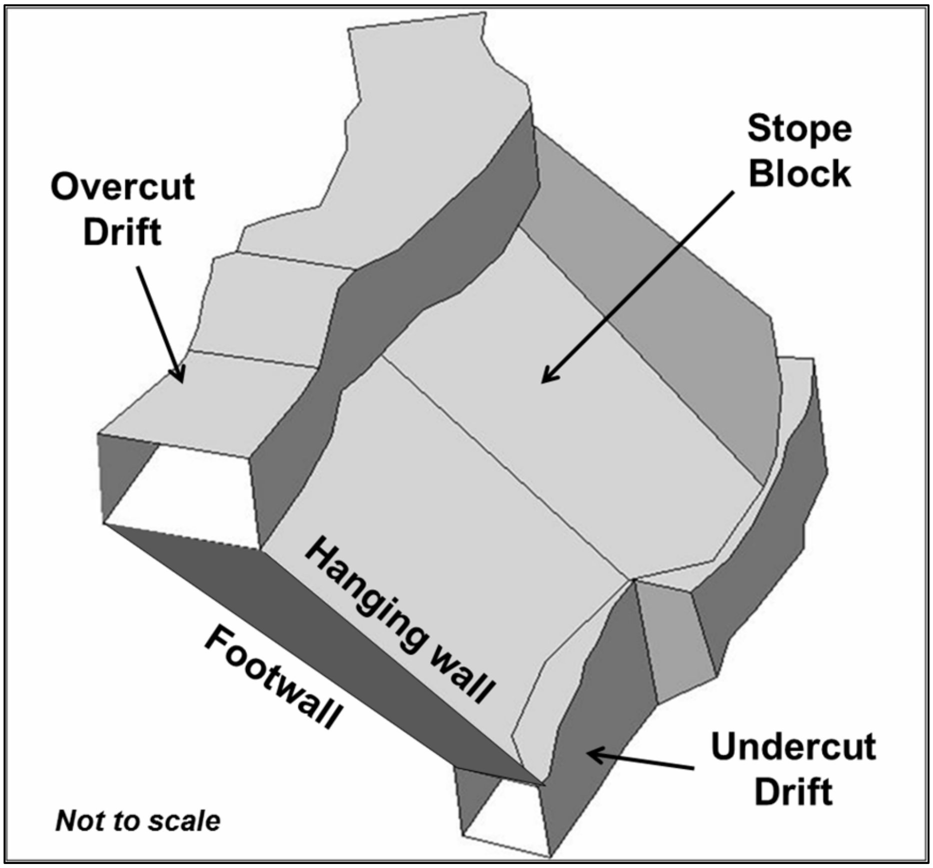


Figure 1-4 Isometric view illustrating the stope overcut drift, undercut drift, hanging wall, and footwall (After Forster et al., 2007).

1.3 Objectives

The main engineering purpose to rock mass classification at Eagle Point Mine is the estimation of open stope dilution. Increased dilution generally translates to increased costs associated with additional material handling, processing, and tailings disposal. A reasonable estimate of dilution is very important for the economic prospects of a mine and developing tools to predict this dilution is a key focus of this thesis.

To enable improved prediction of mining dilution, an appropriate assessment of the rock mass strength properties is required. The rock mass associated with the Eagle Point Mine is heavily sheared and altered, and characterizing the properties of the rock mass for rock mass classification is challenging. The mine has many sources of data describing different aspects of the properties of the rock mass hosting the Eagle Point orebodies. Combining disparate sources of data describing rock mass properties forms an important component of this research.

The mine currently uses a subjective approach for assessing stope geometry both during the design and reconciliation phases. One of the primary objectives was to standardize the stope measurements in order to compare the performances of each of the stopes.

Another objective of this project was to update the current rock mass classification values to the geological categorization system, and develop a simple method of incorporating the exploration diamond drill hole data into rock mechanics design.

Although the data and application for this project are specific to the Eagle Point Mine, the methodology of quantifying stope geometries is applicable to other open stope mining operations. A new method of relating classification systems was developed, and this method may also be used for other rock mechanics applications.

1.4 Scope

The scope of this research will be limited to the rock mass classification systems which are used at the Eagle Point Mine. This thesis focuses on the engineering rock mass properties and stope geometries which influence the amount of stope dilution. Other factors which influence stope dilution are not evaluated.

Point load tests were completed by the author on exploration diamond drill hole core, and the results were compared for trends in the strength by rock type and degree of alteration.

Rock mass classification data from a variety of sources, including ground control audits, site personnel rock mass classification observations, and geological mapping completed by the site geological department were compiled for this project. The data were provided by Cameco Corporation, and analyzed by the author by comparing the rock mass classification values and geological rock types and assessment of rock strength and alteration.

Twenty-seven stope reconciliation case histories were analyzed by the author for the purposes of this project.

1.5 Overview

The method used for the ore extraction at the Eagle Point Mine is the open stope mining method. A significant amount of ore dilution from unstable waste rock, corresponding to the weak rock mass conditions found at this mine, is associated with this mining method. The prediction and prevention of stope dilution is important for controlling mining costs. Appropriate rock mass characterization is crucial for predicting the engineering behaviour of a rock mass.

A complex rock mass, such as is found at the Eagle Point Mine, is difficult to characterize. There are several sources of data available at the mine site to aid in rock mass classification. The rock classification values are applied to current stope dilution prediction methods.

The following chapters introduce rock mass classification methods used at the Eagle Point Mine, dilution prediction methods, improvements to the current mine classification system, and provide open stope case studies using the updated rock mass characterization methods.

Empirical design methods are used for dilution prediction, as they are based on previous underground opening performance. Reliable estimates of rock mass properties are needed to use these empirical methods, as empirical methods rely on representative estimates of rock mass classification. The following chapter presents common rock mass classification methods which are used at the Eagle Point Mine.

Chapter 2 - Literature Review of Rock Mass Classification Systems

This chapter introduces published work related to rock mass classification systems. There are many different systems, and the system used at any particular mine largely depends on the site-specific engineering applications the classification systems are applied to. The scope of this research will be limited to those systems which are used at the Eagle Point Mine. Essentially, this focuses the research on systems suitable for weak rock masses.

The general goal of these rock mass classification systems is to provide an estimate of the rock mass properties of a particular rock mass. The rock mass properties can then be combined with excavation geometry and loading conditions to obtain a prediction of the behaviour of the excavation.

2.1 Data Required for Rock Mass Classification

A rock mass consists of intact rock which contains geological structures including joints, bedding, shear planes, etc. Rock mass classification includes an assessment of the intact rock and the discontinuities within the rock mass.

Most rock mass classification systems use a combination of the following parameters for assessing a rock mass:

1. Intact rock strength
2. Number and orientation of joint sets
3. Joint spacing and Rock Quality Designation (RQD)
4. Joint surface characteristics
5. Groundwater conditions

Joint set orientations, infilling, amplitude, roughness, and spacing will influence the behaviour of a rock mass under loading. This information has been the basis for many classification systems such as the Rock Mass Rating (RMR) system (Bieniawski, 1976), and the Rock Tunnelling

Quality Index (Q) system (Barton et al., 1974). The information may be collected from surface mapping, underground drift mapping or from diamond drill hole information. For this project, drift mapping and data from diamond drill holes are pertinent.

2.1.1 **Intact Rock Strength**

The intact rock strength of a geological unit is the amount of external loading that a sample of that geological unit can withstand before breaking. There are several general methods for measuring the intact rock strength, including laboratory testing, impact methods, and simple field tests.

The compressive strength of a material is its capacity to withstand axially applied forces. The unconfined compressive strength (UCS) of rocks and soils can be determined using laboratory or field tests as defined by the American Society for Testing and Materials (ASTM) and the International Society for Rock Mechanics (ISRM), including the following:

- ASTM D7012 - 10 (ASTM, 2012) Standard Test Method for Compressive Strength and Elastic Moduli of Intact Rock Core Specimens under Varying States of Stress and Temperatures; Method C: Uniaxial Compressive Strength of Intact Rock Core Specimens.
- International Society of Rock Mechanics (ISRM, 1985) Commission on Testing Methods, Suggested Method for Determining Point Load Strength

Jennings and Robertson (1969) proposed several classes of soil and rock strength. The classes, descriptions, and approximate UCS are shown in Table 2-1. Five classes of soils and rocks are defined as S1 through S5, and R1 through R5, respectively. The soils range in strength up to 1 MPa, and the rocks have strengths from 1 MPa to greater than 200 MPa.

Table 2-1 Approximate classification of cohesive soil and rock (After Jennings and Robertson, 1969)

No.	Description	Uniaxial Compressive Strength (MPa)	Examples
S1	VERY SOFT SOIL - easily moulded with fingers, shows distinct heel marks	0.04	
S2	SOFT SOIL - moulds with strong pressure from fingers, shows faint heel marks	0.04 - 0.08	
S3	FIRM SOIL - very difficult to mould with fingers, indented with finger nail, difficult to cut with hand spade	0.08 - 0.15	
S4	STIFF SOIL - cannot be moulded with fingers, cannot be cut with hand spade, requires hand picking for excavation	0.15 - 0.60	
S5	VERY STIFF SOIL - very tough, difficult to move with hand pick, pneumatic spade required for excavation	0.6 - 1.0	
R1	VERY WEAK ROCK - crumbles under sharp blows with geological pick point, can be cut with pocket knife	1 - 25	Chalk, rock salt
R2	MODERATELY WEAK ROCK - shallow cuts or scraping with pocket knife with difficulty, pick point indents deeply with firm blow	25 - 50	Coal, schist, siltstone
R3	MODERATELY STRONG ROCK - knife cannot be used to scrape or peel surface, shallow indentations under firm blow from pick point	50 - 100	Sandstone, slate, shale
R4	STRONG ROCK - hand-held sample breaks with one firm blow from hammer end of geological pick	100 - 200	Marble, granite, gneiss
R5	VERY STRONG ROCK - requires many blows from geological pick to break intact sample	> 200	Quartzite, dolerite, gabbro, basalt

2.1.2 **Joint Set Delineation**

Intact rock strength is important for determining the overall rock mass strength. However, the degree of fracturing, which controls the shape and intact rock blocks that make up the rock mass, is often a controlling factor. The first step in determining the geometry of intact blocks is estimating the number of joint sets. This is done through a structural data gathering method, as shown in Figure 2-1. This figure illustrates an engineer, geologist or technician using a compass to collect dip and dip direction for a joint. An example of a field data collection sheet for the joint dips, dip directions, and descriptions is shown. The data collected could then be entered into a stereonet program to determine pole clusters of joints to determine joint sets. The clusters can be used to estimate or calculate representative orientations for each joint set.

At the Eagle Point Mine, a Silva compass is used to collect joint dip and dip direction. These values are typically collected by Geology staff for each development advance, but may also be collected by the Engineering Department for specific types of analyses. When the data is collected by the Engineering Department, the Rocscience program Dips™ is used for stereonet plotting and analysis.

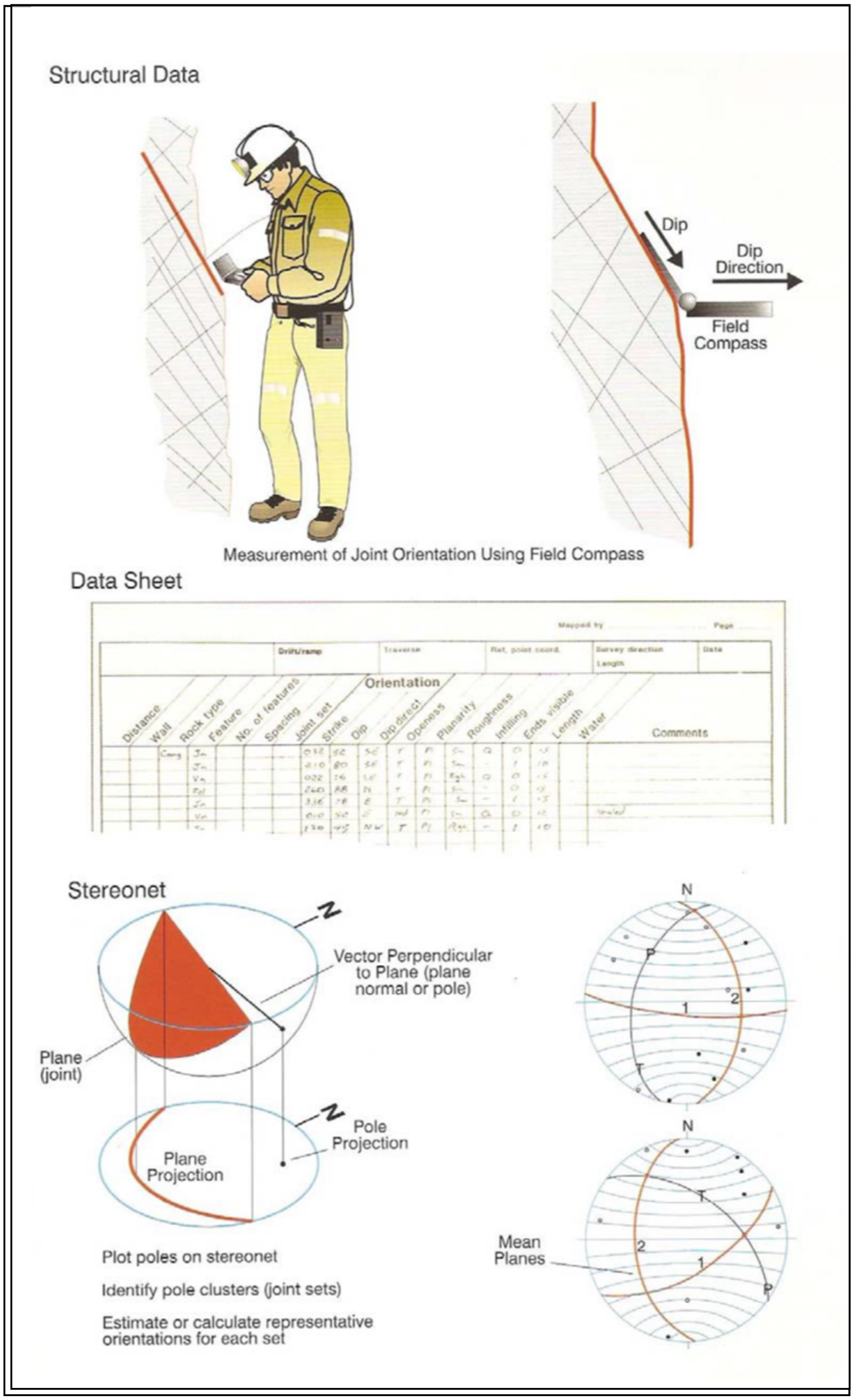


Figure 2-1 Structural data collection (After Hutchinson and Diederichs, 1996, From Nedin and Potvin, 2003).

2.1.3 **Number and Orientation of Joint Sets**

A joint set refers to a group of fracture planes which generally have the same orientation and dip. It is expected that there may be some deviation in the orientation or dip from joint surface to joint surface, but the overall grouping should be similar. Each group would be considered one set, and the total number of sets controls the block shape of the intact blocks of rock that make up the rock mass.

In Figure 2-2, three joint sets are illustrated, and are described with dip and dip directions as follows:

- Joint Set A (yellow) – 70/315
- Joint Set B (red) – 55/045
- Joint Set C (green) – 65/180

Together, these joint sets have created a blocky rock mass with an average block size of about 0.3 m by 0.3 m by 0.3 m (0.027 m^3).

Figure 2-2 Example of number of joint sets; Three joint sets shown (After Cameco Corp. Internal Report, 2012).

2.1.4 **Joint Spacing**

The joint spacing refers to the average perpendicular distance between fracture surfaces. The joint spacing controls the block size for a rock mass. Figure 2-3 shows a 50mm to 500mm joint spacing. These joint spacings are typical for the Eagle Point Mine.

Figure 2-3 Example of joint spacing; 50 – 500mm+ (After Cameco Corp. Internal Report, 2012).

2.1.5 **The Rock Quality Designation (RQD)**

A key component of both the Rock Mass Rating (RMR) (Bieniawski, 1976 and 1989) (described in section 2.2.1), and the Tunnelling Quality Index (Q) (Barton et al., 1974) (described in section 2.2.2) is the Rock Quality Designation index (Deere, 1964). The Rock Quality Designation index, or RQD, was developed in order to use information from drill core to provide a preliminary estimation of rock mass quality. It is defined as the percentage of intact core pieces longer than 100 mm (4 inches) in the total length of core drilled in a core run (Figure 2-4).

Hudson and Priest (1979) have presented the following, mathematical relation equation between RQD and fracture frequency:

$$RQD = 100e^{-0.1\lambda}(1+0.1\lambda)$$

(Equation 2.1)

where,

λ = the average joint frequency

RQD is a directionally dependent parameter, so different drill hole orientations through the same rock mass may return highly variable rock mass property values.

When a rock exposure is available for mapping and rock classification, it is not necessary to rely upon core data. The following formula was proposed by Palmström (1982) to estimate the RQD when discontinuities are visible in the rock mass:

$$RQD = 115 - 3.3\ J_v$$

(Equation 2.2)

where,

J_v = the volumetric joint count, or sum of the number of joints per metre for all discontinuity sets.

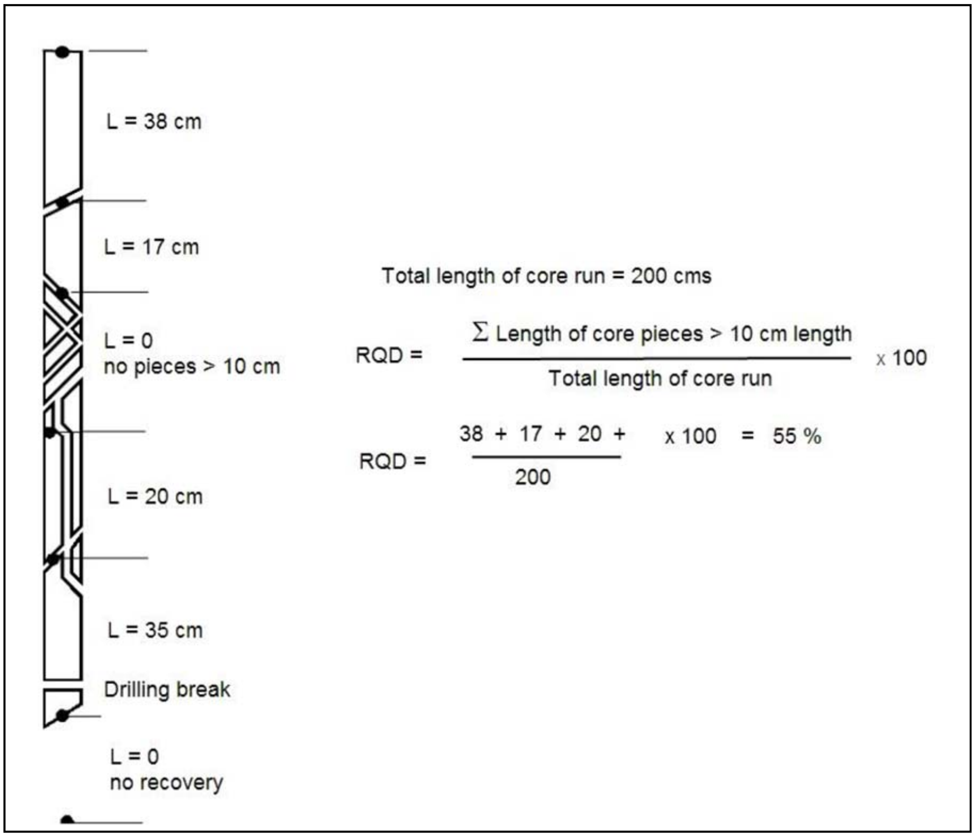


Figure 2-4 Procedure for measurement and calculation of RQD (after Deere, 1989).

2.1.6 **Joint Surface Characteristics**

Joint surface characteristics are important to the overall rock mass strength because they may be an indication of how easily intact blocks of rock will slide past one another. The friction and cohesion of the joint surfaces are dependent on the following individual properties:

- Continuity of joints
- Length of joints
- Basic friction on joint surfaces
- Roughness at varied scales:
    - Slickensided / polished (very small scale)
    - Smooth / rough (10cm scale)
    - Planar / wavy (1m scale of roughness)

Figure 2-5 shows a method of estimating joint roughness by the use of a contour gauge. A contour gauge is constructed of steel pins for accurate matching of contours and shapes and is typically used for carpentry. For joint roughness estimation, the contour gauge is pressed against a joint surface and the resulting profile is compared to the rock joint roughness profiles showing the typical range of Joint Roughness Coefficients (JRC) (Barton and Choubey, 1977) as shown in Figure 2-6. A JRC of less than 10 is considered to be smooth, and a JRC of greater than 10 is assumed to be rough.

Figure 2-5 Contour gauge tool used for estimating joint roughness coefficient (After Capes, 2009).

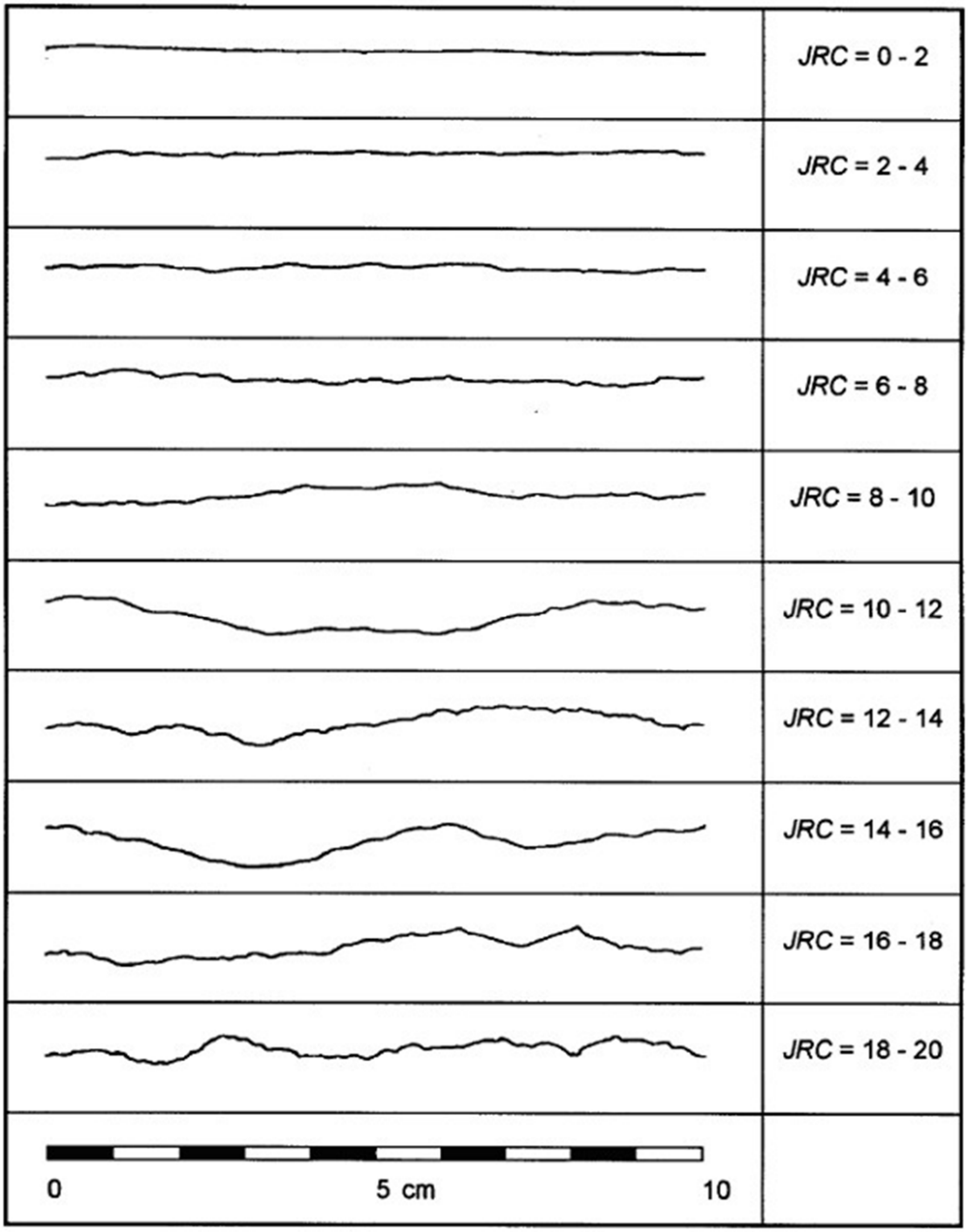


Figure 2-6 Rock joint roughness profiles showing the typical range of joint roughness coefficients (JRC) (Barton and Choubey, 1977).

2.1.7 **Groundwater condition**

Water flowing through underground fractures may act as a lubricant to reduce the frictional properties of the fracture surfaces, or destabilize blocks of rock with increased water pressures. Thus, it is important to note the approximate flow of water along the joint surfaces. The Eagle Point Mine has hydrogeological zones ranging from dry to wet, or 0 – 25 Litres/min per 10 m of exposed tunnel. In isolated areas, there are zones which have inflow rates of greater than 25 Litres/min.

2.2 Rock Mass Classification Systems

There are a variety of rock mass classification systems which use some or all of the rock mass properties discussed. These systems have generally been designed to be simple tools for estimating the quality of a rock mass and are often linked to easily used empirical design techniques. Although each system uses different parameters and weightings for the characterization of the rock mass, the common factor is that most use multi-parameter classification schemes which assess the overall rock mass properties. Section 2.2 presents three main systems which will be examined for the purpose of this thesis. These are the two Rock Mass Rating (RMR) systems (Bieniawski, 1976 and 1989), and the Tunnelling Quality Index (Q) (Barton et al., 1974). These are discussed because they are widely accepted, and are used in empirical design methods which will be presented in Chapter 3.

2.2.1 **RMR (Rock Mass Rating) Classification**

The Rock Mass Rating (RMR) system was developed by Bieniawski (1976) and was later revised to reflect changes in the weighting of the ratings assigned to different parameters (Bieniawski, 1989). For both the RMR_{76} and the revised RMR_{89} classifications, the following six main parameters of a rock mass are assessed:

1. UCS of the rock material. The UCS of the rock mass may be measured in a laboratory, or estimated using field techniques.
2. RQD. The RQD of the rock mass may be measured from drill core, or may be estimated in an underground or surface exposure.
3. Spacing of discontinuities
4. Condition of discontinuities
5. Groundwater conditions
6. Orientation of discontinuities

The rock mass is given a numerical rating based upon the above six parameters, and the sum total denotes the RMR. Table 2-2 shows the range of values assigned to each parameter as per the RMR_{76} ratings.

The most notable change from RMR_{76} to RMR_{89} is a decrease in the total weighting value of the spacing of the joints, and an increase in the total weighting of both the condition of joints and groundwater parameters. Table 2-3 shows the range of values assigned to each parameter as per the RMR_{89} ratings.

In addition to updating the weightings assigned to each parameter, the RMR_{89} revision also expanded the descriptions for the condition of joints, thus accounting for a larger variety of joint surface conditions. The guidelines for classification of discontinuity conditions may be found in Table 2-4. Furthermore, RMR_{89} can account for the dip and dip direction of joint sets in relation to the direction of tunnelling, as shown in Table 2-5 and Table 2-6.

Table 2-2 Rock Mass Rating system classification parameters and their ratings (After Bieniawski, 1976)

<table>
<tr><th colspan="3">PARAMETER</th><th colspan="7">RANGES OF VALUES</th></tr>
<tr><td rowspan="3">1</td><td rowspan="2">Strength of intact rock material</td><td>Point Load Strength index</td><td>> 8 MPa</td><td>4 - 8 MPa</td><td>2 - 4 MPa</td><td>1 - 2 MPa</td><td colspan="3">For this low range uniaxial compressive strength test is preferred</td></tr>
<tr><td>Uniaxial compressive strength</td><td>> 200 MPa</td><td>100 - 200 MPa</td><td>50 - 100 MPa</td><td>25 - 50 MPa</td><td>10 - 25 MPa</td><td>3 - 10 MPa</td><td>1 - 3 MPa</td></tr>
<tr><td colspan="2">Rating</td><td>15</td><td>12</td><td>7</td><td>4</td><td>2</td><td>1</td><td>0</td></tr>
<tr><td rowspan="2">2</td><td colspan="2">Drill Core quality, RQD</td><td>90% - 100%</td><td>75% - 90%</td><td>50% - 75%</td><td>25% - 50%</td><td colspan="3">< 25%</td></tr>
<tr><td colspan="2">Rating</td><td>20</td><td>17</td><td>13</td><td>8</td><td colspan="3">3</td></tr>
<tr><td rowspan="2">3</td><td colspan="2">Spacing of Joints</td><td>> 3 m</td><td>1 - 3 m</td><td>0.3 - 1 m</td><td>50 - 300 mm</td><td colspan="3">< 50 mm</td></tr>
<tr><td colspan="2">Rating</td><td>30</td><td>25</td><td>20</td><td>10</td><td colspan="3">5</td></tr>
<tr><td rowspan="2">4</td><td colspan="2">Condition of Joints</td><td>Very rough surfaces
Not continuous
No separation
Hard joint wall rock</td><td>Slightly rough surfaces
Separation < 1 mm
Hard joint wall rock</td><td>Slightly rough surfaces
Separation < 1 mm
Soft joint wall rock</td><td>Slickensided surfaces
or
Gouge < 5 mm thick
or
Joints open 1 - 5 mm
Continuous joints</td><td colspan="3">Soft gouge > 5 mm thick
or
Joints open > 5 mm
Continuous joints</td></tr>
<tr><td colspan="2">Rating</td><td>25</td><td>20</td><td>12</td><td>6</td><td colspan="3">0</td></tr>
<tr><td rowspan="4">5</td><td rowspan="3">Groundwater</td><td>Inflow per 10 m tunnel length</td><td colspan="2">None</td><td>< 25 litres/min</td><td>25 - 125 litres/min</td><td colspan="3">> 125 litres/min</td></tr>
<tr><td>Ratio (joint water pressure/major principal stress)</td><td colspan="2">0</td><td>0.0 - 0.2</td><td>0.2 - 0.5</td><td colspan="3">> 0.5</td></tr>
<tr><td>General Conditions</td><td colspan="2">Completely dry</td><td>Moist only (interstitial water)</td><td>Water under moderate pressure</td><td colspan="3">Severe water problems</td></tr>
<tr><td colspan="2">Rating</td><td colspan="2">10</td><td>7</td><td>4</td><td colspan="3">0</td></tr>
</table>

Table 2-3 Rock Mass Rating system classification parameters and their ratings (After Bieniawski, 1989)

	PARAMETER		RANGES OF VALUES						
1	Strength of intact rock material	Point Load Strength index	> 10 MPa	4 - 10 MPa	2 - 4 MPa	1 - 2 MPa	For this low range uniaxial compressive strength test is preferred		
		Uniaxial compressive strength	> 250 MPa	100 - 250 MPa	50 - 100 MPa	25 - 50 MPa	5 - 25 MPa	1 - 5 MPa	< 1 MPa
	Rating		15	12	7	4	2	1	0
2	Drill Core quality, RQD		90% - 100%	75% - 90%	50% - 75%	25% - 50%	< 25%		
	Rating		20	17	13	8	3		
3	Spacing of Joints		> 2 m	0.6 - 2 m	200 - 600 mm	60 - 200 mm	< 60 mm		
	Rating		20	15	10	8	5		
4	Condition of Joints		Very rough surfaces Not continuous No separation Unweathered wall rock	Slightly rough surfaces Separation < 1 mm Slightly weathered walls	Slightly rough surfaces Separation < 1 mm Highly weathered walls	Slickensided surfaces or Gouge < 5 mm thick or Joints open 1 - 5 mm Continuous joints	Soft gouge > 5 mm thick or Separation > 5 mm Continuous joints		
	Rating		30	25	20	10	0		
5	Groundwater	Inflow per 10 m tunnel length	None	< 10 litres/min	10 - 25 litres/min	25 - 125 litres/min	> 125 litres/min		
		Ratio (joint water pressure/major principal stress)	0	< 0.1	0.1 - 0.2	0.2 - 0.5	> 0.5		
		General Conditions	Completely dry	Damp	Wet	Dripping	Flowing		
	Rating		15	10	7	4	0		

Table 2-4 Guidelines for classification of discontinuity conditions (After Bieniawski, 1989)

Discontinuity Length (persistence)	< 1 m	1 - 3 m	3 - 10 m	10 - 20 m	> 20 m
Rating	6	4	2	1	0
Separation (aperture)	None	< 0.1 mm	0.1 - 1.0 mm	1 - 5 mm	> 5 mm
Rating	6	5	4	1	0
Roughness	Very rough	Rough	Slightly rough	Smooth	Slickensided
Rating	6	5	3	1	0
Infilling (gouge)	None	Hard filling < 5 mm	Hard filling > 5 mm	Soft filling < 5 mm	Soft filling > 5 mm
Rating	6	4	2	2	0
Weathering	Unweathered	Slightly weathered	Moderately weathered	Highly weathered	Decomposed
Rating	6	5	3	1	0

Table 2-5 Effect of discontinuity strike and dip orientation in tunnelling (Modified after Wickham et al., 1972)

<table>
<tr><td colspan="4">Strike Perpendicular to tunnel axis</td><td colspan="2" rowspan="2">Strike parallel to tunnel axis</td><td rowspan="3">Dip 0° - 20° irrespective of strike</td></tr>
<tr><td colspan="2">Drive with dip</td><td colspan="2">Drive against dip</td></tr>
<tr><td>Dip 45° - 90°</td><td>Dip 20° - 45°</td><td>Dip 45° - 90°</td><td>Dip 20° - 45°</td><td>Dip 45° - 90°</td><td>Dip 20° - 45°</td></tr>
<tr><td>Very favourable</td><td>Favourable</td><td>Fair</td><td>Unfavourable</td><td>Very Unfavourable</td><td>Fair</td><td>Unfavourable</td></tr>
</table>

Table 2-6 Rock Mass Rating adjustment for discontinuity orientations (After Bieniawski, 1989)

Strike and dip orientations of joints		Very favourable	Favourable	Fair	Unfavourable	Very unfavourable
Ratings	Tunnels & Mines	0	-2	-5	-10	-12
	Foundations	0	-2	-7	-15	-25
	Slopes	0	-5	-25	-50	

2.2.2 **Norwegian Geotechnical Institute Classification System, Q and Modified Q, Q'**

The Norwegian Geotechnical Institute (NGI) proposed another method of classifying rock masses. The Rock Mass Tunnelling Quality Index (Q) (Barton et al., 1974) is calculated using the following formula:

$$Q = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times \frac{J_w}{SRF}$$

(Equation 2.3)

where,

RQD = Rock Quality Designation

J_n = joint set number

J_r = joint roughness number

J_a = joint alteration number

J_w = joint water reduction factor

SRF = stress reduction factor

The first quotient (RQD/J_n) roughly correlates to the block or particle size, while the second quotient (J_r/J_a) is an estimation of the roughness and frictional characteristics of the fracture surfaces and infilling material. The overall Q value will increase with larger block sizes, fewer joint sets, and rough, clean joint surfaces. Conversely, small block sizes, many joint sets, and clay infillings will decrease the Q value. The third quotient (J_w/SRF) consists of two stress parameters corresponding to stresses acting on the rock. Table 2-7 (After Barton et al., 1974) shows the classification categories and values for calculating the Tunnelling Quality Index, Q, for a rock mass.

Table 2-7 Classification of individual parameters in the Tunnelling Quality Index, Q (After Barton et al., 1974)

DESCRIPTION	VALUE	NOTES
1. ROCK QUALITY DESIGNATION	***RQD***	
A. Very poor	0 - 25	1. Where *RQD* is reported or measured as ≤ 10 (including 0),
B. Poor	25 - 50	a nominal value of 10 is used to evaluate *Q*.
C. Fair	50 - 75	
D. Good	75 - 90	2. *RQD* intervals of 5, i.e. 100, 95, 90 etc. are sufficiently
E. Excellent	90 - 100	accurate.
2. JOINT SET NUMBER	J_n	
A. Massive, no or few joints	0.5 - 1.0	
B. One joint set	2	
C. One joint set plus random	3	
D. Two joint sets	4	
E. Two joint sets plus random	6	
F. Three joint sets	9	1. For intersections use (3.0 × J_n)
G. Three joint sets plus random	12	
H. Four or more joint sets, random, heavily jointed, 'sugar cube', etc.	15	2. For portals use (2.0 × J_n)
J. Crushed rock, earthlike	20	
3. JOINT ROUGHNESS NUMBER	J_r	
a. Rock wall contact		
b. Rock wall contact before 10 cm shear		
A. Discontinuous joints	4	
B. Rough and irregular, undulating	3	
C. Smooth undulating	2	
D. Slickensided undulating	1.5	1. Add 1.0 if the mean spacing of the relevant joint set is
E. Rough or irregular, planar	1.5	greater than 3 m.
F. Smooth, planar	1.0	
G. Slickensided, planar	0.5	2. J_r = 0.5 can be used for planar, slickensided joints having
c. No rock wall contact when sheared		lineations, provided that the lineations are oriented for
H. Zones containing clay minerals thick enough to prevent rock wall contact	1.0 (nominal)	minimum strength.
J. Sandy, gravely or crushed zone thick enough to prevent rock wall contact	1.0 (nominal)	
4. JOINT ALTERATION NUMBER	J_a	ϕr degrees (approx.)
a. Rock wall contact		
A. Tightly healed, hard, non-softening, impermeable filling	0.75	1. Values of ϕr, the residual friction angle, are intended as an approximate guide
B. Unaltered joint walls, surface staining only	1.0	25 - 35 to the mineralogical properties of the
C. Slightly altered joint walls, non-softening mineral coatings, sandy particles, clay-free disintegrated rock, etc.	2.0	25 - 30 alteration products, if present.
D. Silty-, or sandy-clay coatings, small clay-fraction (non-softening)	3.0	20 - 25
E. Softening or low-friction clay mineral coatings, i.e. kaolinite, mica. Also chlorite, talc, gypsum and graphite etc., and small quantities of swelling clays. (Discontinuous coatings, 1 - 2 mm or less)	4.0	8 - 16

Table 2-7 (Cont'd.) Classification of individual parameters in the Tunnelling Quality Index, Q (After Barton et al., 1974)

4, JOINT ALTERATION NUMBER	J_a	ϕr degrees (approx.)	
b. Rock wall contact before 10 cm shear			
F. Sandy particles, clay-free, disintegrating rock etc.	4.0	25 - 30	
G. Strongly over-consolidated, non-softening clay mineral fillings (continuous < 5 mm thick)	6.0	16 - 24	
H. Medium or low over-consolidation, softening clay mineral fillings (continuous < 5 mm thick)	8.0	12 - 16	
J. Swelling clay fillings, i.e. montmorillonite, (continuous < 5 mm thick). Values of J_a depend on percent of swelling clay-size particles, and access to water.	8.0 - 12.0	6 - 12	
c. No rock wall contact when sheared			
K. Zones or bands of disintegrated or crushed	6.0		
L. rock and clay (see G, H and J for clay	8.0		
M. conditions)	8.0 - 12.0	6 - 24	
N. Zones or bands of silty- or sandy-clay, small clay fraction, non-softening	5.0		
O. Thick continuous zones or bands of clay	10.0 - 13.0		
P. & R. (see G.H and J for clay conditions)	6.0 - 24.0		
5. JOINT WATER REDUCTION	J_w	approx. water pressure (kgf/cm^2)	
A. Dry excavation or minor inflow i.e. < 5 l/m locally	1.0	< 1.0	
B. Medium inflow or pressure, occasional outwash of joint fillings	0.66	1.0 - 2.5	
C. Large inflow or high pressure in competent rock with unfilled joints	0.5	2.5 - 10.0	1. Factors C to F are crude estimates; increase J_w if drainage installed.
D. Large inflow or high pressure	0.33	2.5 - 10.0	
E. Exceptionally high inflow or pressure at blasting, decaying with time	0.2 - 0.1	> 10	2. Special problems caused by ice formation are not considered.
F. Exceptionally high inflow or pressure	0.1 - 0.05	> 10	
6. STRESS REDUCTION FACTOR		*SRF*	
a. Weakness zones intersecting excavation, which may cause loosening of rock mass when tunnel is excavated			
A. Multiple occurrences of weakness zones containing clay or chemically disintegrated rock, very loose surrounding rock any depth)		10.0	1. Reduce these values of *SRF* by 25 - 50% but only if the relevant shear zones influence do not intersect the excavation
B. Single weakness zones containing clay, or chemically dis-tegrated rock (excavation depth < 50 m)		5.0	
C. Single weakness zones containing clay, or chemically dis-tegrated rock (excavation depth > 50 m)		2.5	
D. Multiple shear zones in competent rock (clay free), loose surrounding rock (any depth)		7.5	
E. Single shear zone in competent rock (clay free). (depth of excavation < 50 m)		5.0	
F. Single shear zone in competent rock (clay free). (depth of excavation > 50 m)		2.5	
G. Loose open joints, heavily jointed or 'sugar cube', (any depth)		5.0	

Table 2-7 (Cont'd.) Classification of individual parameters in the Tunnelling Quality Index, Q (After Barton et al., 1974)

DESCRIPTION	VALUE			NOTES
6. STRESS REDUCTION FACTOR			*SRF*	
b. Competent rock, rock stress problems				
	σ_c/σ_1	$\sigma_t\sigma_1$		2. For strongly anisotropic virgin stress field (if measured): when $5 \leq \sigma_1/\sigma_3 \leq 10$, reduce σ_c to $0.8\sigma_c$ and σ_t to $0.8\sigma_t$. When $\sigma_1/\sigma_3 > 10$, reduce σ_c and σ_t to $0.6\sigma_c$ and $0.6\sigma_t$, where σ_c = unconfined compressive strength, and σ_t = tensile strength (point load) and σ_1 and σ_3 are the major and minor principal stresses.
H. Low stress, near surface	> 200	> 13	2.5	
J. Medium stress	200 - 10	13 - 0.66	1.0	
K. High stress, very tight structure (usually favourable to stability, may be unfavourable to wall stability)	10 - 5	0.66 - 0.33	0.5 - 2	
L. Mild rockburst (massive rock)	5 - 2.5	0.33 - 0.16	5 - 10	
M. Heavy rockburst (massive rock)	< 2.5	< 0.16	10 - 20	3. Few case records available where depth of crown below surface is less than span width. Suggest *SRF* increase from 2.5 to 5 for such cases (see H).
c. Squeezing rock, plastic flow of incompetent rock under influence of high rock pressure				
N. Mild squeezing rock pressure			5 - 10	
O. Heavy squeezing rock pressure			10 - 20	
d. Swelling rock, chemical swelling activity depending on presence of water				
P. Mild swelling rock pressure			5 - 10	
R. Heavy swelling rock pressure			10 - 15	

ADDITIONAL NOTES ON THE USE OF THESE TABLES

When making estimates of the rock mass Quality (Q), the following guidelines should be followed in addition to the notes listed in the tables:

1. When borehole core is unavailable, *RQD* can be estimated from the number of joints per unit volume, in which the number of joints per metre for each joint set are added. A simple relationship can be used to convert this number to *RQD* for the case of clay free rock masses: $RQD = 115 - 3.3 J_v$ (approx.), where J_v = total number of joints per m^3 ($0 < RQD < 100$ for $35 > J_v > 4.5$).
2. The parameter J_n representing the number of joint sets will often be affected by foliation, schistosity, slaty cleavage or bedding etc. If strongly developed, these parallel 'joints' should obviously be counted as a complete joint set. However, if there are few 'joints' visible, or if only occasional breaks in the core are due to these features, then it will be more appropriate to count them as 'random' joints when evaluating J_n.
3. The parameters J_r and J_a (representing shear strength) should be relevant to the weakest significant joint set or clay filled discontinuity in the given zone. However, if the joint set or discontinuity with the minimum value of J_r/J_a is favourably oriented for stability, then a second, less favourably oriented joint set or discontinuity may sometimes be more significant, and its higher value of J_r/J_a should be used when evaluating Q. The value of J_r/J_a should in fact relate to the surface most likely to allow failure to initiate.
4. When a rock mass contains clay, the factor *SRF* appropriate to loosening loads should be evaluated. In such cases the strength of the intact rock is of little interest. However, when jointing is minimal and clay is completely absent, the strength of the intact rock may become the weakest link, and the stability will then depend on the ratio rock-stress/rock-strength. A strongly anisotropic stress field is unfavourable for stability and is roughly accounted for as in note 2 in the table for stress reduction factor evaluation.
5. The compressive and tensile strengths (σ_c and σ_t) of the intact rock should be evaluated in the saturated condition if this is appropriate to the present and future in situ conditions. A very conservative estimate of the strength should be made for those rocks that deteriorate when exposed to moist or saturated conditions.

2.2.3 **Comparison of Rock Mass Rating, RMR, and Rock Mass Tunnelling Quality Index, Q**

The Rock Mass Rating systems (Bieniawski, 1976 and 1989) and the Rock Mass Tunnelling Quality Index, Q (Barton et al., 1974) systems have similar input parameters, but it is difficult to directly compare the two systems. In Figure 2-7, Bieniawski (1989) plotted a range of Q values with their equivalent RMR_{89} ratings, and proposed the following equation to relate the two systems.

$$RMR_{76} = 9 \ln Q + 44$$

(Equation 2.4)

This relationship is not exact, as can be seen from the range of data in Figure 2-7. This is due to the difference in weightings for the parameters in each of the systems, as well as the parameters considered. The RMR system assesses rock strength and joint orientation, while the Q system considers the number of joint sets and not the rock strength or joint orientation.

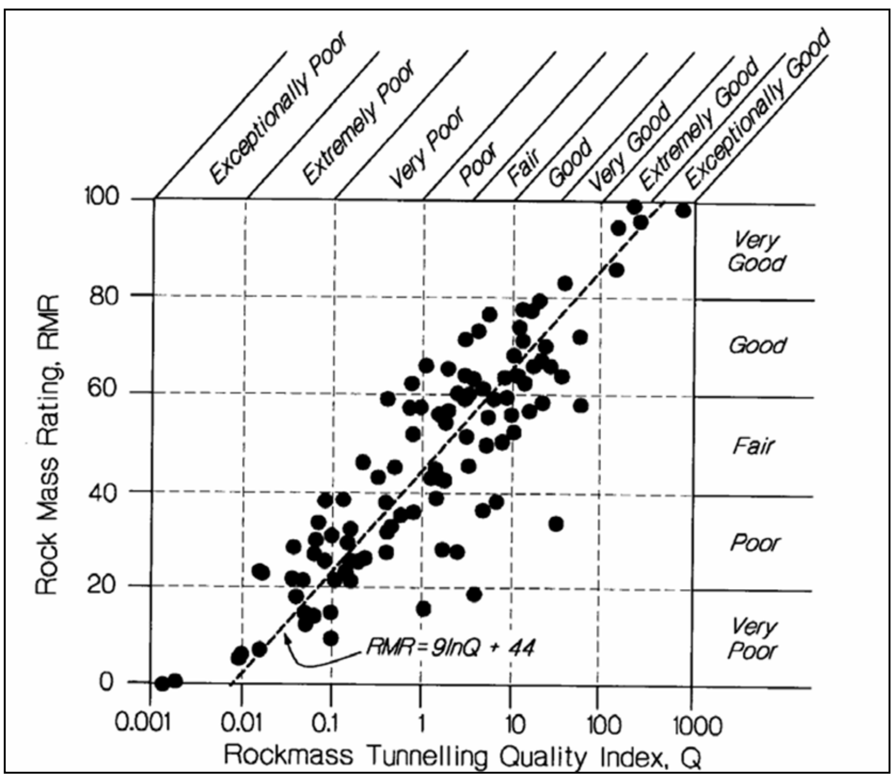


Figure 2-7 Suggested relationship between RMR_{76} and Q (from Bieniawski, 1989).

2.3 Summary

All three of these classification systems are applied at the Eagle Point Mine. The external Rock Mechanics consultant for the mine uses the RMR_{76} system for rock mass classification, and site personnel use the RMR_{89} system for rock mass classification, and the Q system for ground support design. Although there are similarities between all of the systems, there are important differences with the input data, as well as the applications the classification systems have been used for. This is described further in the following chapter.

Chapter 3 – Literature Review of Empirical Span, Open Stope Stability, and Dilution Design Methods

This section introduces empirical design techniques commonly used in mining to determine if failure can be expected, and in some cases, predict the degree of failure, often expressed as dilution. Most rock mechanics design methods consider the opening geometry, the loading conditions, and some method of rock mass classification for predicting the stability, or the degree of instability, of the rock mass.

Open stopes are considered non-entry locations for personnel, so the design criteria for stability is less conservative than areas which require access for personnel. For example, main access drifts, ramps, emergency refuge stations, and underground shop facilities require a more conservative design due to the length of time these areas will be used, and the number of personnel routinely present.

Both numerical and empirical methods exist for designing underground openings, and both approaches include a component of rock mass classification. Numerical design methods generally use rock mass classification values as input parameters for stress-based ground failure criteria. Empirical design methods use classification to compare similar rock mass conditions, and generally exist as graphical representations based on stable and failed ground openings (Milne, 2007).

In general, empirical design methods for rock mass properties have been the most effective in weak rock masses because they permit the overall behaviour of a rock mass to be predicted easily and accurately, while allowing for changes to be made based on observed mining conditions (Brady et al., 2005). The basis for the success of an empirical method is having adequate field data of good quality coupled with ongoing field observations that allow changing rock conditions to be evaluated as mining progresses. Some of the more commonly used empirical design methods used for mining are presented.

3.1 Empirical Span Design Method

The Span Curve (Lang, 1994) was developed using the RMR_{76} classification and the span of an opening. Figure 3-1 shows the Span Design Curve that was updated by Wang et al. (2002) to include 292 observations of locally supported spans which were unstable, potentially unstable, or stable. The span of the opening on the y-axis is the diameter of the largest circle that can fit between the walls or pillar in an opening. The design method only applies to horizontal backs of drifts or stopes. The RMR_{76} value is plotted on the x-axis. In cases where the critical joint set dips at less than 30 degrees, from Table 2-5 and Table 2-6, 10 is subtracted from the RMR_{76} value to account for structure. If evidence of high stress is apparent, 10 is again subtracted from the RMR_{76} value. The stable and unstable zones were determined from the case histories of openings and their performance, while the potentially unstable zone includes a combination of case studies which may or may not have been stable. The plot of span and RMR_{76} helps to predict the potential stability of an opening.

This graph is very useful for assessing the potential instability of a back opening, but cannot be used for open stope walls.

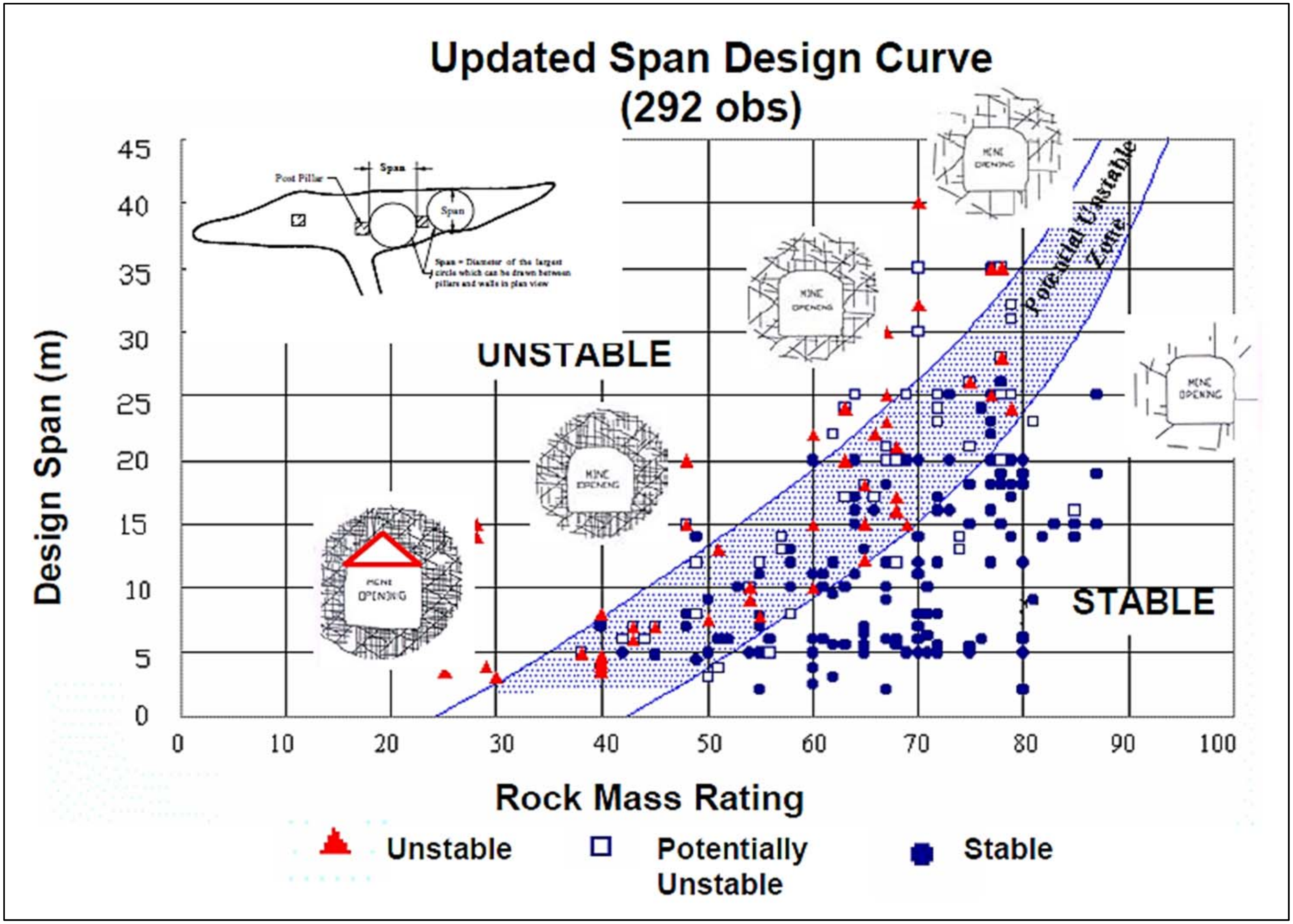


Figure 3-1 Critical span curve for mine entry methods employing local support only (Wang et al., 2002).

3.2 Stability Graph

Mathews et al. (1981) developed an empirical method for open stope design at depths below 1,000 m. The initial stability chart was based on a relatively small set of data and case studies. Potvin (1988) included many more case studies, and proposed several factors for stability. The Updated Mathews Stability Graph as proposed by Potvin (1988) is shown in Figure 3-2. The Modified Stability Number (N') is plotted on a logarithmic scale along the y-axis, and the Hydraulic Radius (HR) is plotted along the x-axis. The Modified Stability Number (N') and Hydraulic Radius (HR) factors are described in more detail in the following sections.

There are four zones which are differentiated. The Stable zone is an area which should be stable without support. The Caved zone is an area which will yield and continue to yield for a particular rock quality and geometry. The Support Required zone denotes an area where an opening would be stable with additional support installed. The Transition zone is an area where a rock quality at a particular geometry would either be stable or may cave.

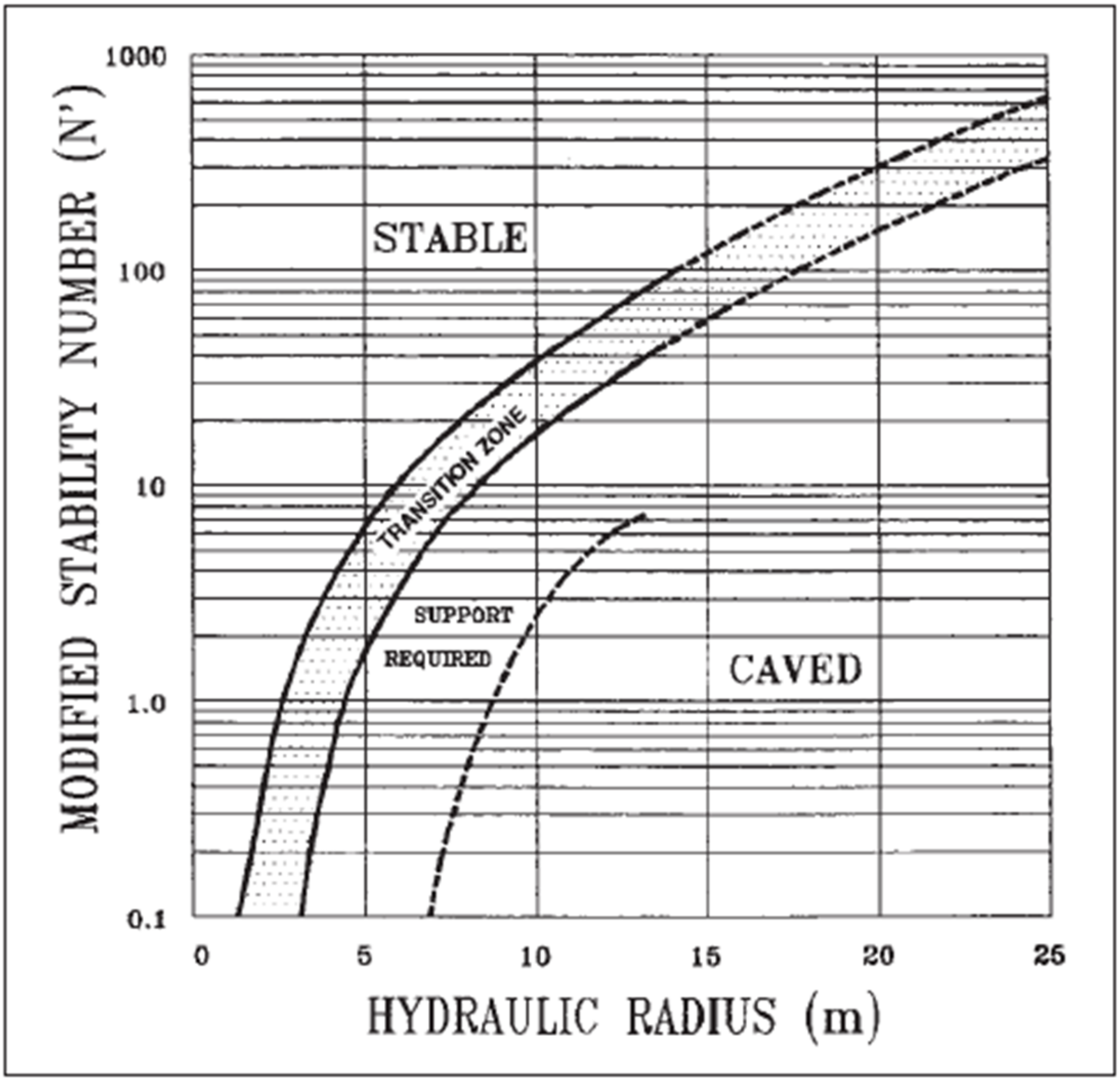


Figure 3-2 Updated Mathews Stability Graph (after Potvin, 1988).

3.2.1 **Modified Stability Number, N'**

The Modified Stability Number, used in the Stability Graph, includes a modified Q value (Q'), as shown in Equation 3.1. The Stress Reduction Factor (SRF) is removed from the Q calculation (Equation 2.3), as it is accounted for by other factors in the Modified Stability Number.

$$Q' = \frac{RQD}{J_n} \times \frac{J_r}{J_a} \times J_w$$

(Equation 3.1)

where,

- RQD = Rock Quality Designation
- J_n = joint set number
- J_r = joint roughness number
- J_a = joint alteration number
- J_w = joint water reduction factor

Figure 3-2 uses the Modified Stability Number, N', which is calculated based upon the following equation:

$$N' = Q' \ x \ A \ x \ B \ x \ C$$

(Equation 3.2)

where,

- Q' = Modified Q value (Equation 3.1)
- A = function of the ratio between the intact rock strength and the induced stress of the excavation.
- B = measure of the relative orientation of the dominant joint set to the excavation surface.
- C = measure of the influence of gravity on the stability of the face being considered.

Potvin (1988) defined the "A" factor as the stress factor for the stope. It is the relationship between the intact rock UCS, and the induced stress in the underground surface being analyzed, and the A factor is designed to account for compressive failure in the rock mass (Figure 3-3).

The UCS may be obtained from laboratory testing as described in section 2.1.1. The maximum induced compressive stress, σ_{max}, may be obtained from two- or three-dimensional stress modeling. For an opening surface that is large relative to the other opening surfaces, such as a hanging wall for a stope, induced stresses near the large opening will be low (Potvin, 1988). This will be represented by an A factor of 1.0, indicating that the large surface is in relaxation or a state of low stress relative to the rock strength.

As the "A" factor accounts for the induced stress of the excavation, the Q system SRF factor as described in section 2.2.2 becomes redundant (Potvin, 1988). Thus, for the purpose of using the Tunnelling Quality Index (Q) for stope design, this value is removed from the equation, resulting in the Q' parameter.

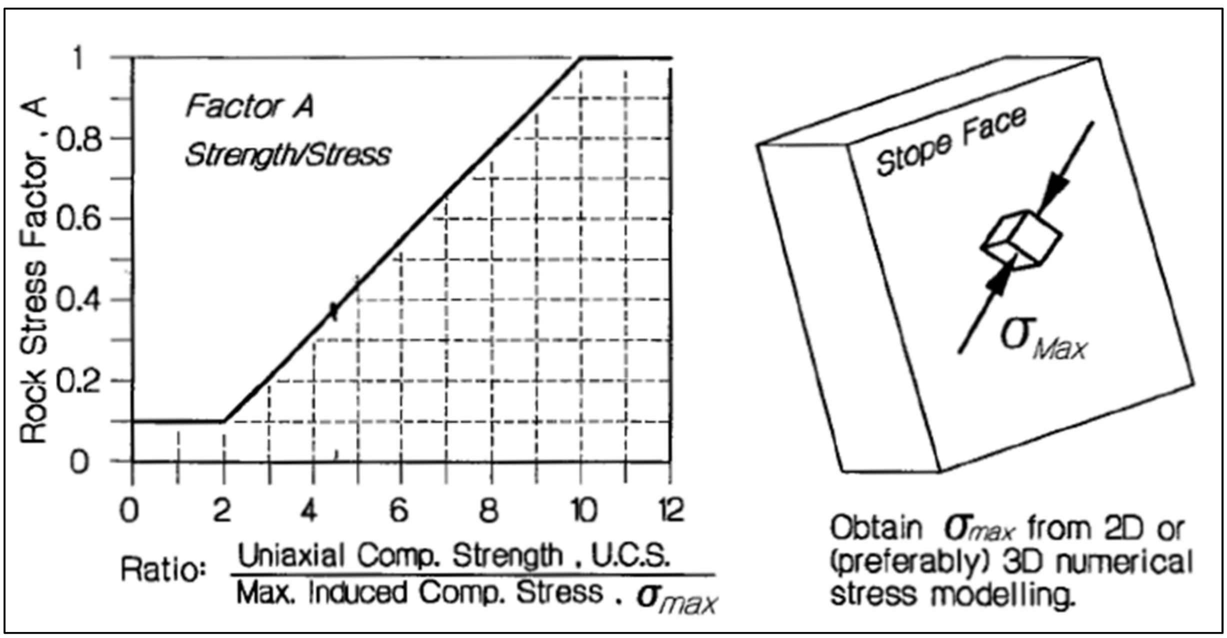


Figure 3-3 Rock Stress Factor, A, for Stability Graph analysis (After Potvin, 1988, from Hutchinson and Diederichs, 1996).

Potvin (1988) defined the “B” factor as the joint orientation factor. This factor accounts for the joint or foliation orientation in relation to the hanging wall surface (Figure 3-4). The true angle between the face and the joint is determined based on the orientations of the major joint sets, and the orientation of the surface being analyzed. The difference between the two angles is plotted along the x-axis, and the B factor is read from the graph. The least stable joint orientation is when joint sets occur sub-parallel to the design surface, or 10° to 30° to the angle of the surface, allowing peeling off along the joint surfaces.

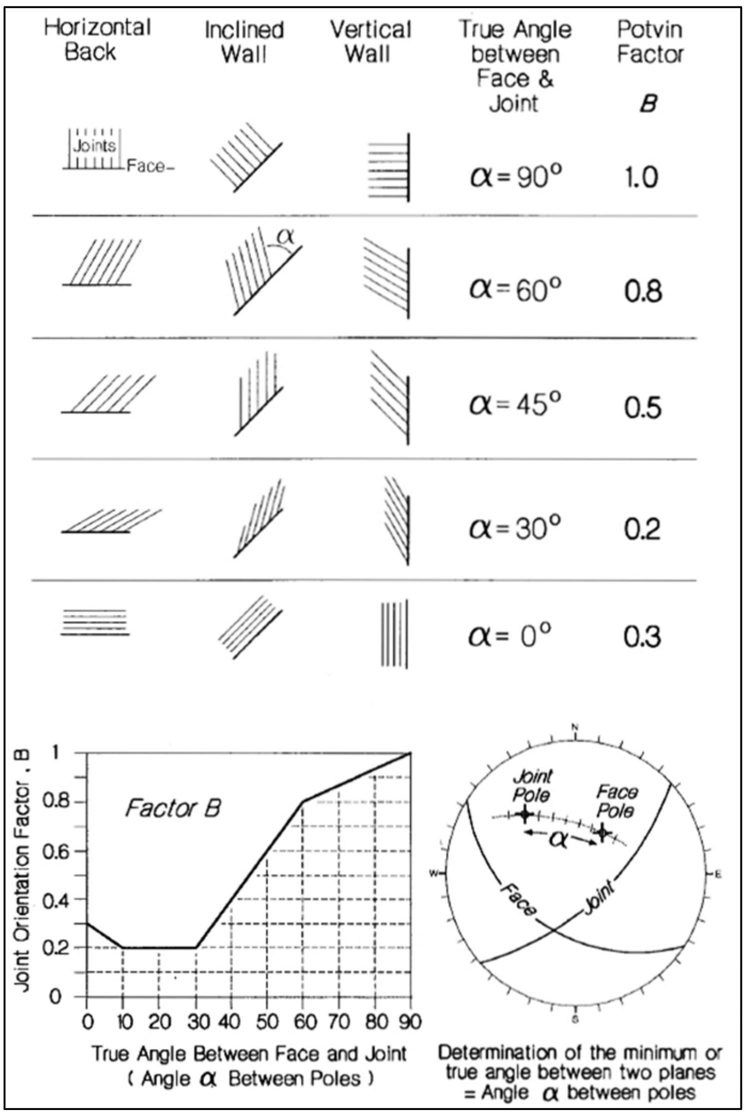


Figure 3-4 Determination of Joint Orientation Factor, B, for Stability Graph analysis (After Potvin, 1988, from Hutchinson and Diederichs, 1996).

Potvin (1988) defined the "C" factor as the stope orientation factor. This represents the effect of gravity on the stability of the stope surface, or face. For gravity fall and slabbing, C may be read from the chart using the dip of the stope face, or calculated using the following formula:

$$C = 8 - 6 \cos(\text{Dip of stope face})$$

(Equation 3.3)

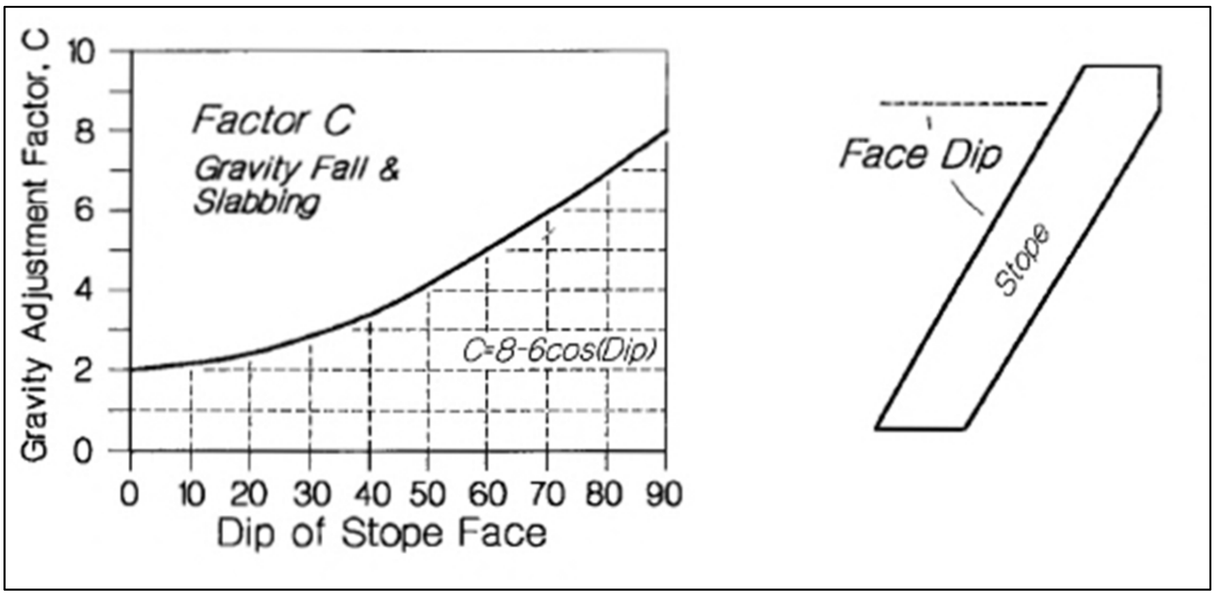


Figure 3-5 Determination of Gravity Adjustment Factor, C, for Stability Graph analysis (After Potvin, 1988, from Hutchinson and Diederichs, 1996).

3.2.2 **Hydraulic Radius**

The hydraulic radius (HR) is the ratio between the surface area of a face of the excavation (m^2) to the perimeter of that face of the excavation (m), as shown in Figure 3-6. For the three-dimensional illustration shown, the "Design Face" is the hanging wall, and the width, w, is the length of the stope, while the height, h, is the up-dip length of the hanging wall. It is possible to calculate the hydraulic radius for each of the exposed surfaces, but the hanging wall and the back of the overcut drift are the two that are commonly a concern for analysis.

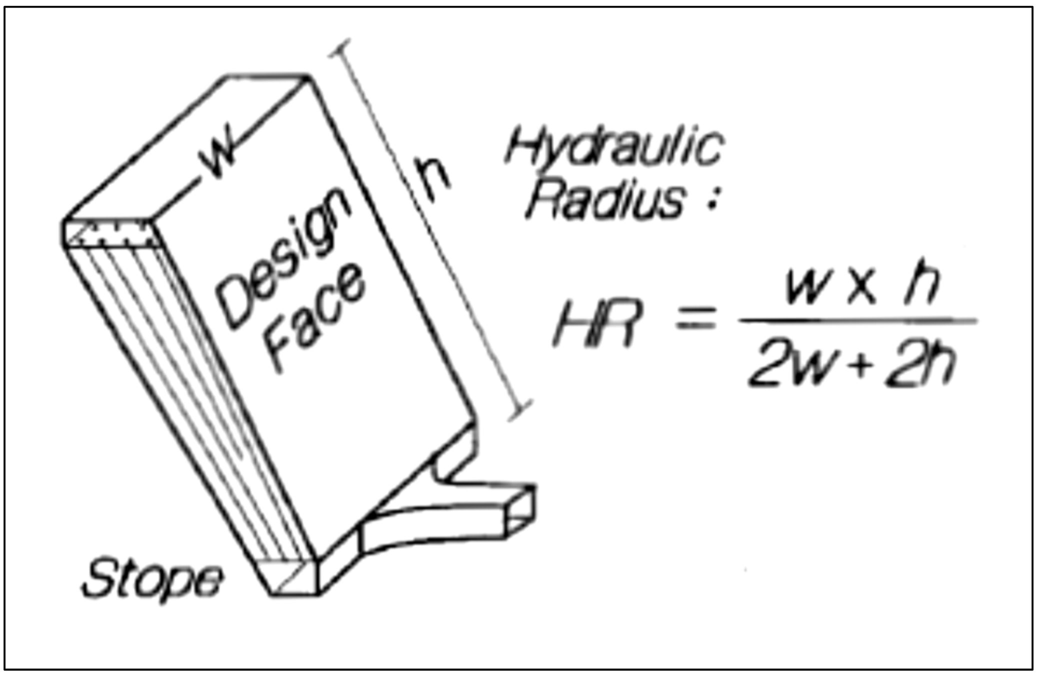


Figure 3-6 Hydraulic radius (HR) from Hutchinson and Diederichs, 1996.

3.3 Dilution Graph Technique

Potvin (1988), Potvin and Milne (1992), and Nickson (1992) collected case histories of unsupported open stopes, and cable bolt supported stopes. These authors identified zones in which stopes were stable or unstable, and a transition zone where some were stable and some were unstable. The Modified Stability Number, N', was plotted against the Hydraulic Radius for each case study. Nickson's updated Stability Graph (1992) is shown in Figure 3-7. Using additional case studies, Nickson was able to better define Potvin's areas of the transition zone (Figure 3-2).

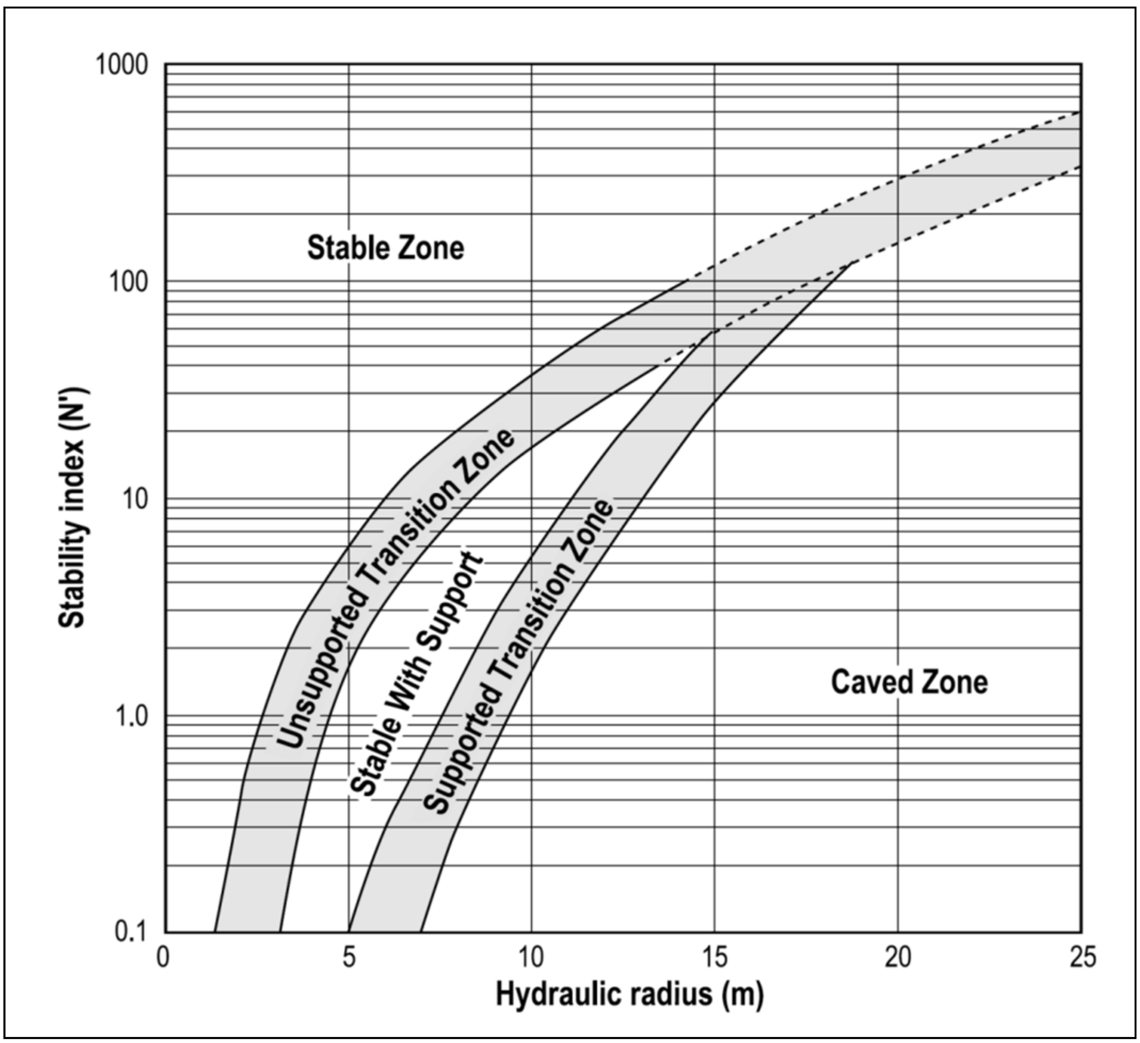


Figure 3-7 Stability Graph (After Nickson, 1992).

3.3.1 **Empirical Dilution Design Methods**

Due to the complexity and anisotropic nature of rock masses, empirical design methods are generally the most effective tools for predicting rock mass behaviour. Clark (1998) used the Modified Stope Graph to create the Dilution Graph (Figure 3-8) which attempts to predict the depth of failure expressed as Equivalent Linear Overbreak/Slough (ELOS) for different Modified Stability Numbers, N', and Hydraulic Radii (HR).

The majority of the case studies used for the development of the Modified Stability Graph were for higher quality rock masses. Pakalnis et al. (2007) with the Geomechanics Group at the University of British Columbia, Canada, in conjunction with the National Institute for Occupational Safety and Health's (NIOSH) Spokane Research Laboratory, developed design guidelines for underground mining within weak rock masses (RMR_{76} under 45 and/or a Q-value under 1.0). Data from the Eagle Point Mine was used to augment the design case histories. Capes (2009) proposed changes to the graph to reflect the analysis of open stopes in weaker rock masses. Figure 3-9 shows the updated graph based on the inclusion of the weak rock mass case studies.

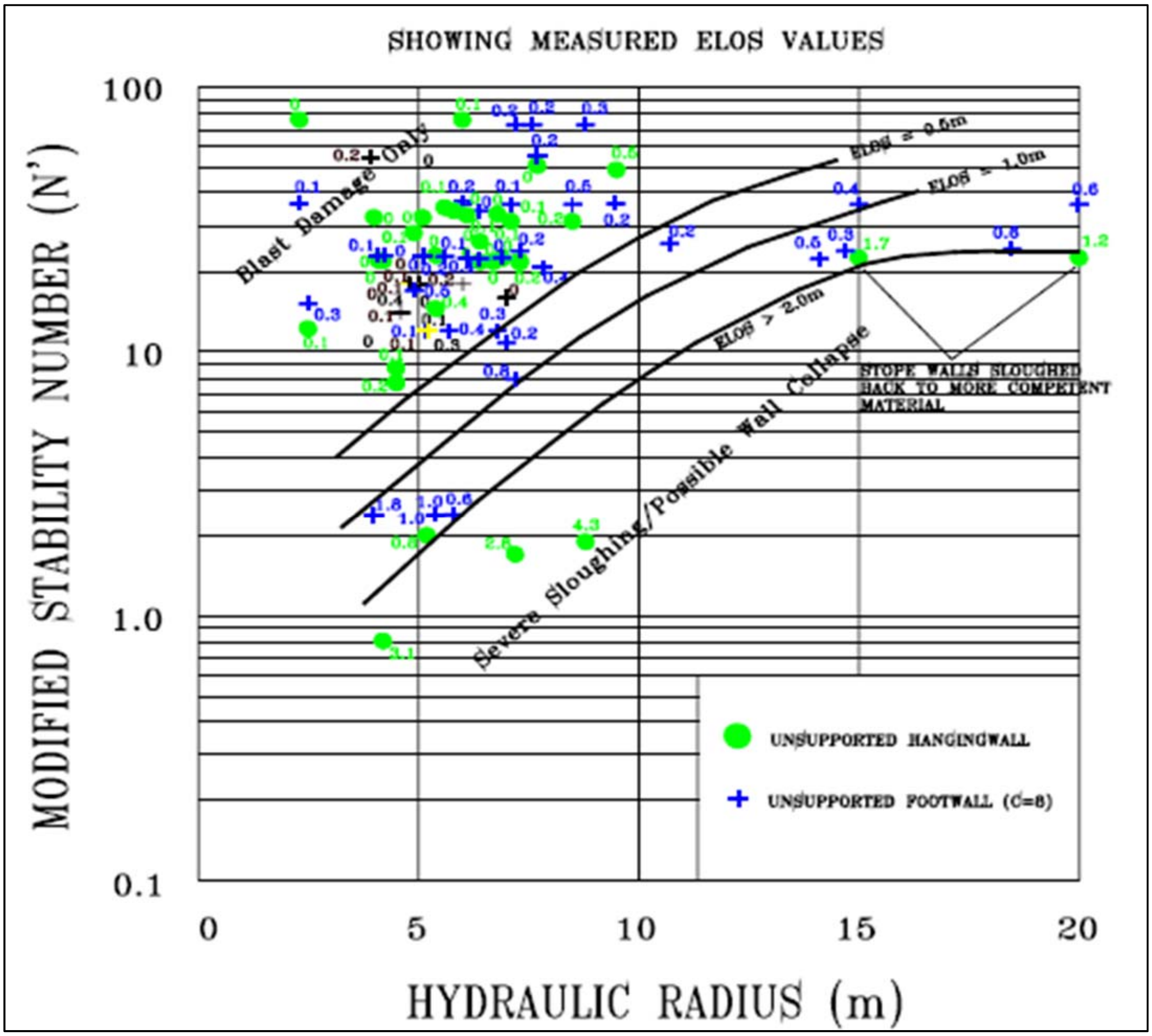


Figure 3-8 Dilution Graph (From Clark, 1998).

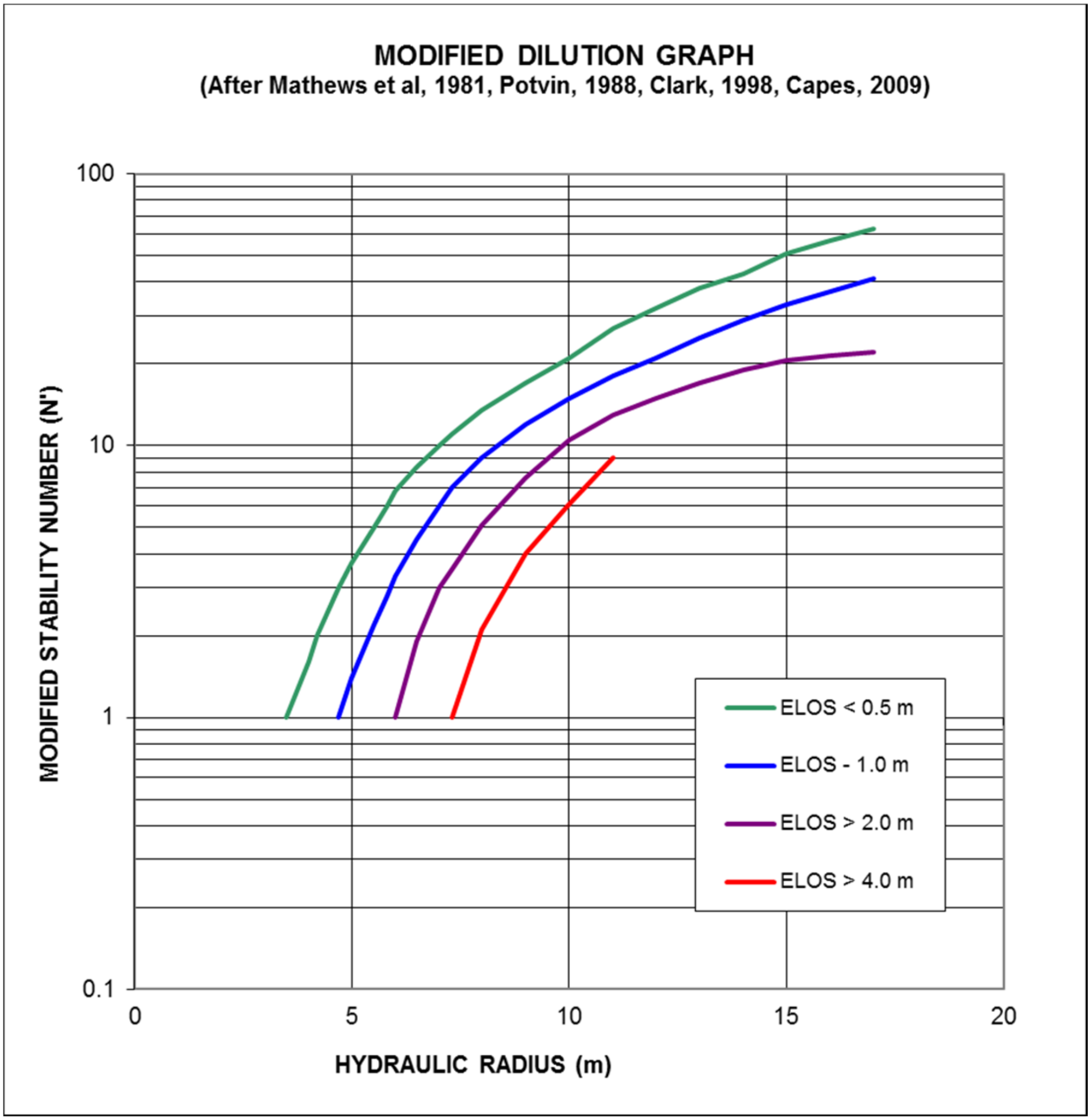


Figure 3-9 Modified Dilution Graph (After Capes, 2009).

3.4 Quantifying Stope Stability and Dilution with the Cavity Monitoring System (CMS)

The effectiveness of the stope design and cable bolt support may be estimated visually, but a more accurate method of assessment is the cavity monitoring survey, CMS (Miller et al., 1992). A CMS is completed after the stope has been mined out, but before it has been backfilled. The CMS equipment consists of a boom with a rotating laser at the end which scans the opening created by the stope, as shown in Figure 3-10. The laser scan of the open stope geometry is compared to the original stope design in order to calculate the dilution, or overbreak, of waste rock that was mined with the stope.

Although the CMS scan allows for a quantitative assessment of the amount of dilution in an opening, there are some limitations to the method. The rotating head is at a fixed point, so the quality of the data is highly dependent on the line-of-site between the instrument and the stope walls.

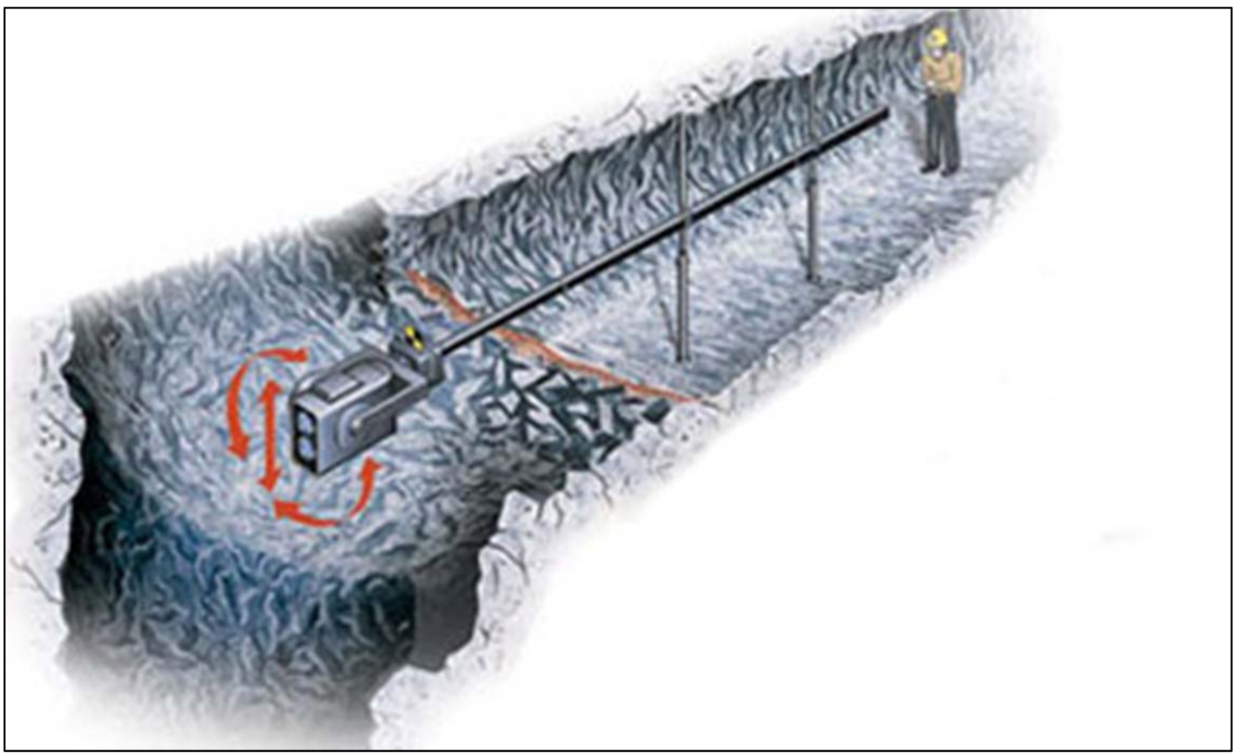


Figure 3-10 Underground set-up and operation of cavity monitoring system (Optech, 2006).

At the Eagle Point Mine, the CMS scans are completed from the overcut drift as there is less likelihood of falling material damaging the equipment. Figure 3-11 illustrates some common problems encountered during data collection. Since the CMS laser records data at set angular increments, there will be a higher density of point data closer to the scanner; areas farther away from the scanner will have less data points and less accuracy. Any protrusions will inhibit the data collection, falsely indicating less overbreak. Material left in the stope will also interfere with the quality of the CMS data.

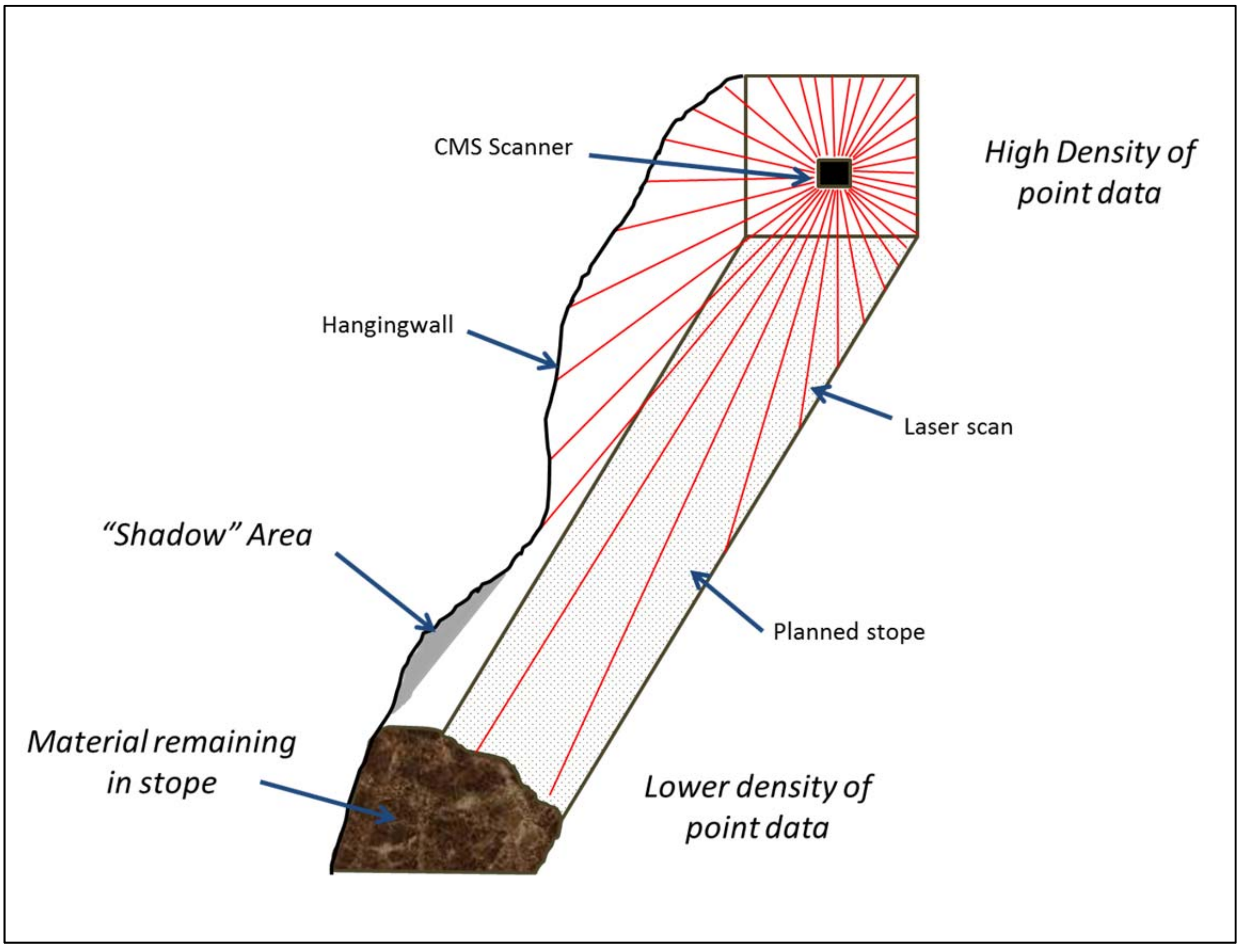


Figure 3-11 Section view of a stope during a CMS scan. Some common difficulties of interpreting CMS scan data are shown.

3.5 Other Factors which Cause Dilution

The empirical stability / dilution graph approach includes many important factors influencing the stability and dilution of an underground opening. Stope size is quantified with the hydraulic radius calculation, and the stope geometry is assessed with the "C" factor for the Stability graph (Potvin, 1988). Other factors used in the stability graph / dilution graph approach quantify the influence of rock mass factors and the effect of induced stresses along stope surfaces. The stability / dilution graphs generally work well in ideal mining conditions, however, many factors can influence rock mass behaviour that are not quantified in these empirical approaches. The following section summarizes some of these factors.

3.5.1 **Undercutting**

Undercutting occurs when material outside of the geology block is removed either intentionally or unintentionally by mining. Figure 3-12 shows a section view example of undercutting in both the overcut and undercut drifts. The development of both drifts occurred into the immediate hanging wall of the stope.

Undercutting can occur during the drift development phase, as shown in Figure 3-12, or from overbreak of stopes below. Wang (2004) stated that undercutting will increase the zone of relaxation for a stope hanging wall. This occurs because an additional free face is developed in the hanging wall, reducing confinement of the immediate hanging wall which allows the rock mass to relax. Undercutting also breaks the beam created by the immediate hanging wall.

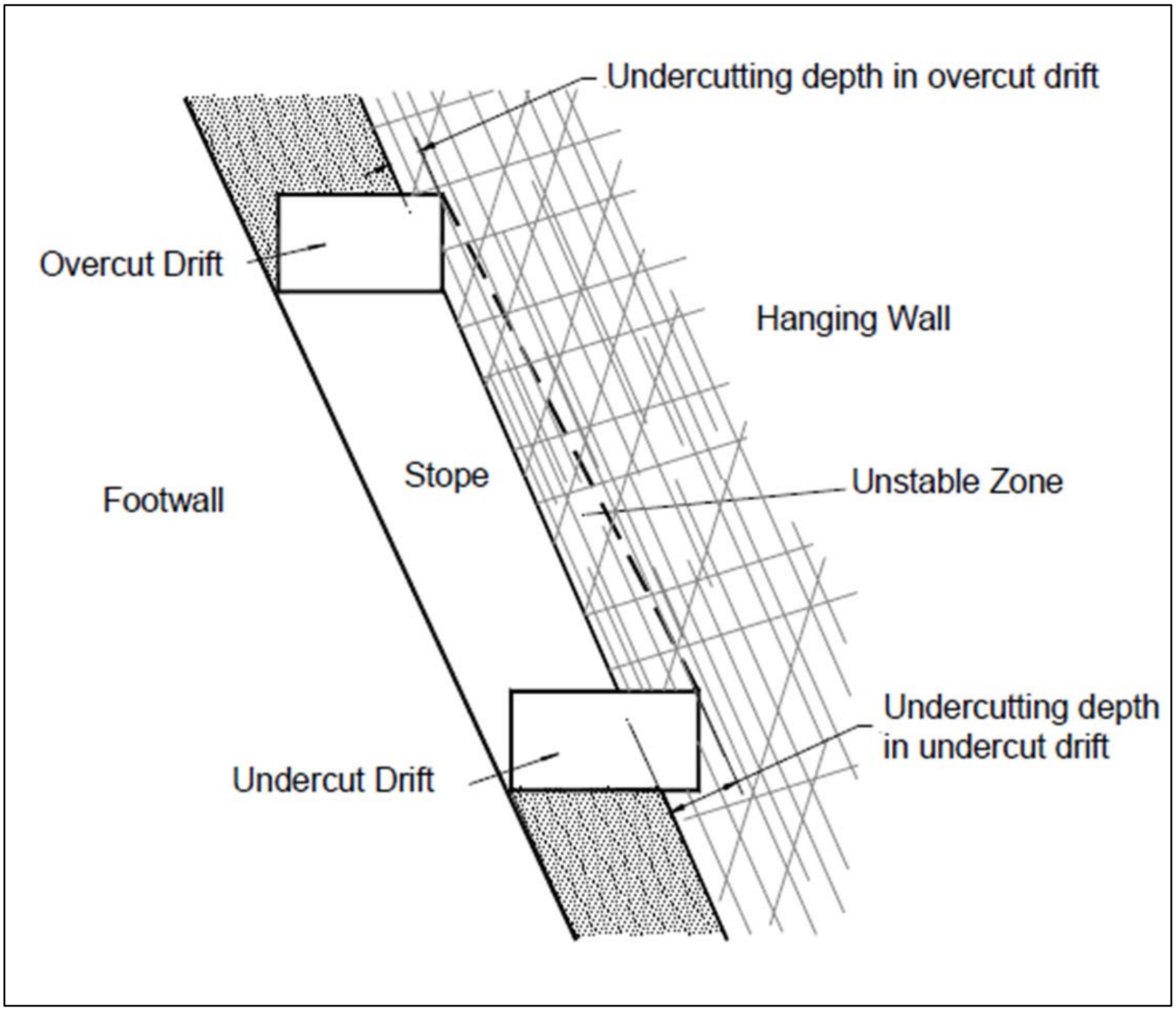


Figure 3-12 Schematic illustration of instability caused by undercutting (Wang, 2004).

3.5.2 **Drilling Practices**

Drill hole deviation is a well-recognized factor contributing to stope dilution at underground mining operations (Yao et al., 1999). Some deviation of the stope drill holes used for blasting is to be expected during stope preparation, but it may vary depending on the driller, drilling equipment, and lithology. Stope drill hole deviation is generally not measured, but may have an effect on the material removed if the holes “wander” too far into the hanging wall (Figure 3-13). Human error is always possible regardless of the precautions taken to prevent it.

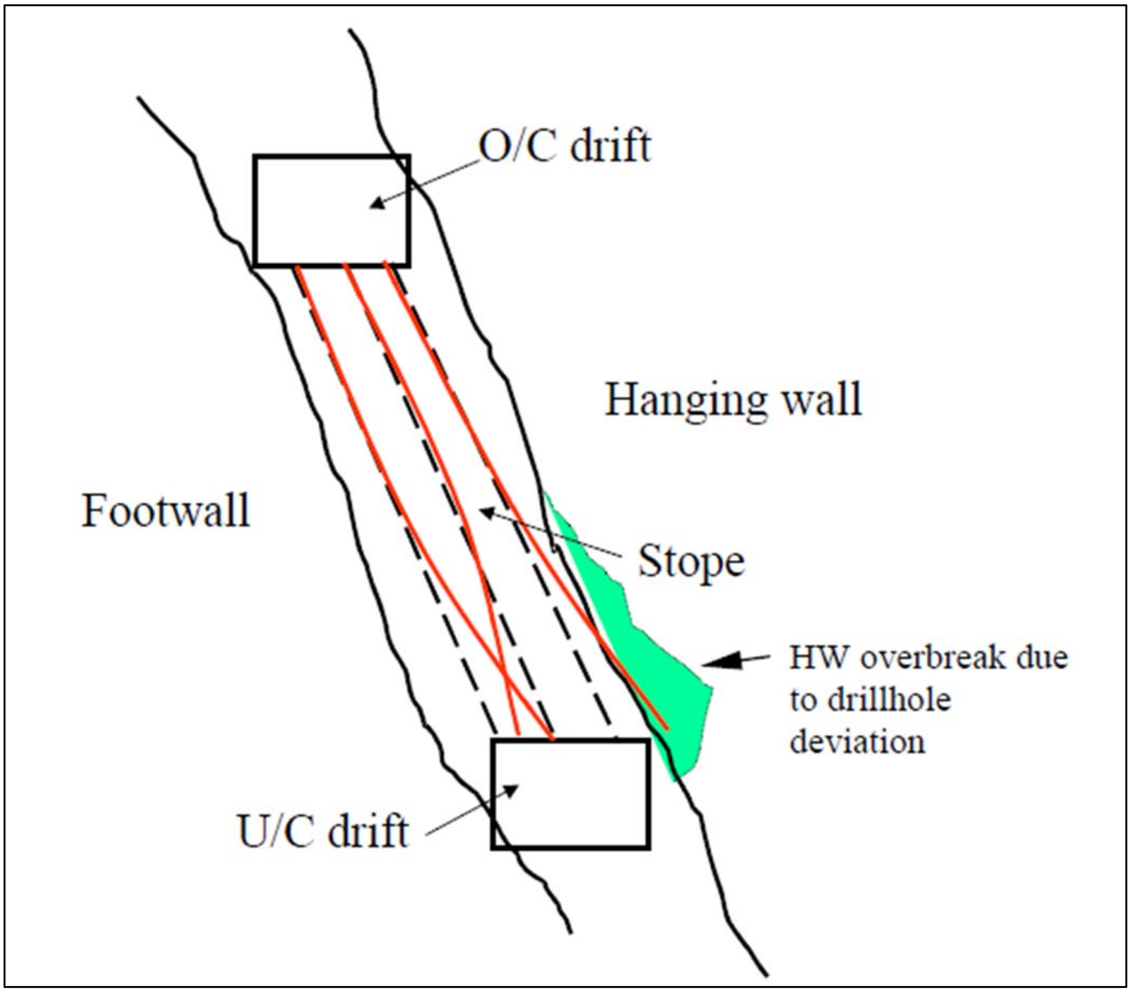


Figure 3-13 Cross section showing theoretical drill hole deviation orientations (From Wang, 2004).

3.5.3 **Blasting Practices**

The type and amount of explosives used per blast may have an effect on the dilution or overbreak of a stope. Blasts are designed to create easily handled fragments of ore. Ideally all of the ore material is removed without damaging the walls beyond the ore/waste contact. However, in order to break the rock mass within the mineralization zone, some damage to the walls is accepted in favor of removing all of the ore. Blast damage may induce new fractures in the rock mass and can also loosen the surrounding rock. This can result in wall instability and increased slough and dilution.

Additionally, vibrations from seismic events, or other blasting or mining activities, may cause more of the rock mass to shake loose from the stope.

3.5.4 **Geological Conditions**

The rock mass is quantified using a classification method, which is limited to the amount of data available from rock exposures and other sources of information. Due to the inhomogeneous nature of rock, conditions may change over relatively short distances. Inferences may be made as to the trend of a structural feature such as a fault, or a shear zone, but it is not possible to know the locations and extents of these features within the rock mass prior to exposure. Projections can be very close to observations in the field, and occasionally may be correlated with CMS data.

The rock mass will often preferentially fail along features such as faults and shears as the rock mass is generally weaker in these areas. If enough geological information is available, the hanging wall stability may be assessed for different zones within the hanging wall until a sufficiently thick and stable zone is intersected by a progressive failure (Capes, 2009). The Modified Stability Graph (Potvin, 1988) does not usually account for discrete faults and/or shear zones within the rock mass, unless these features are sufficiently extensive to affect the overall rock mass rating. In such cases where discrete faults and/or shear zones exist, the opening surface would be better assessed for discrete failure.

Figure 3-14 shows that Stope 245-055 was undercut by the stope below, but the CMS scan also indicates that the stope hanging wall broke to a projected geological feature. In certain circumstances, the rock mass may also be weaker along a lithological contact. These areas are more difficult to predict with increasingly complex structural geology.

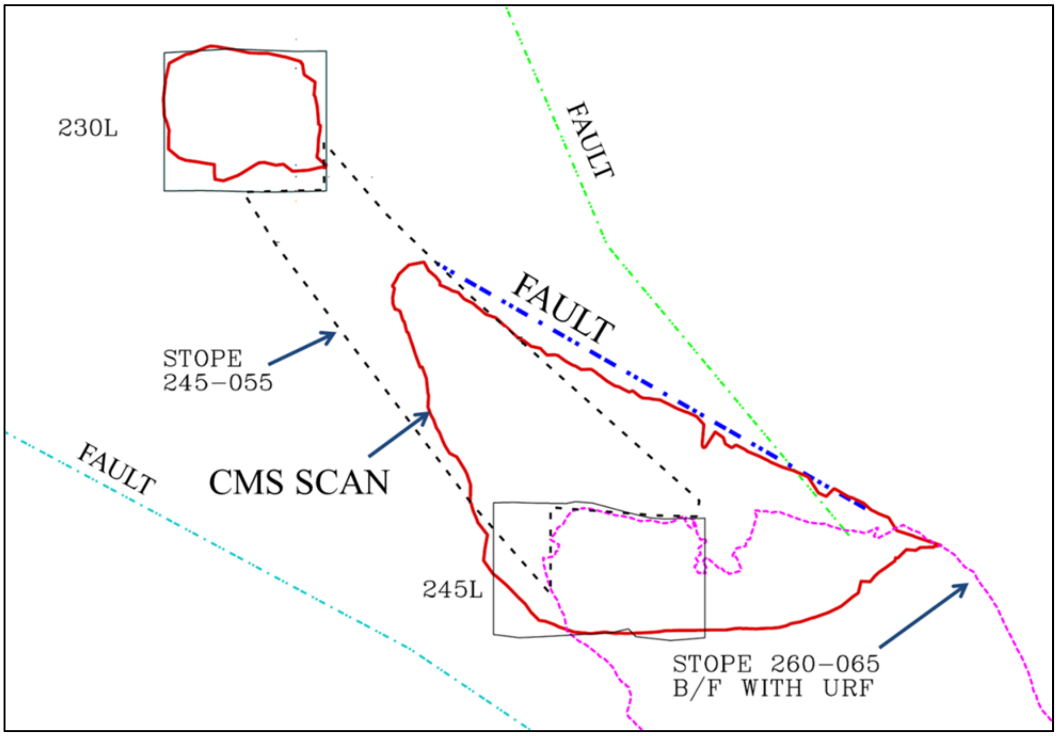


Figure 3-14 Cross section of stope 245-055. Stope 260-065 below (magenta) was backfilled with uncemented rock fill (URF). CMS scan (red) indicates stope hanging wall broke to a geological contact (blue).

3.5.5 **Exposure Time**

Early observations were made by Lauffer (1958) and Bieniawski (1976) showed that tunnel excavation conditions deteriorated over time. Bieniawski (1989) conducted research which suggests that the effect of the length of exposure time increases as the quality of the rock mass decreases. The RMR system was used to develop a graph relating RMR and unsupported stand up time, as shown in Figure 3-15.

As with underground tunnels, over time, ground relaxation can cause an increase in the amount of dilution of a stope. This has been studied by Wang et al. (2003) and Violot et al. (2012), however, results have not been conclusive.

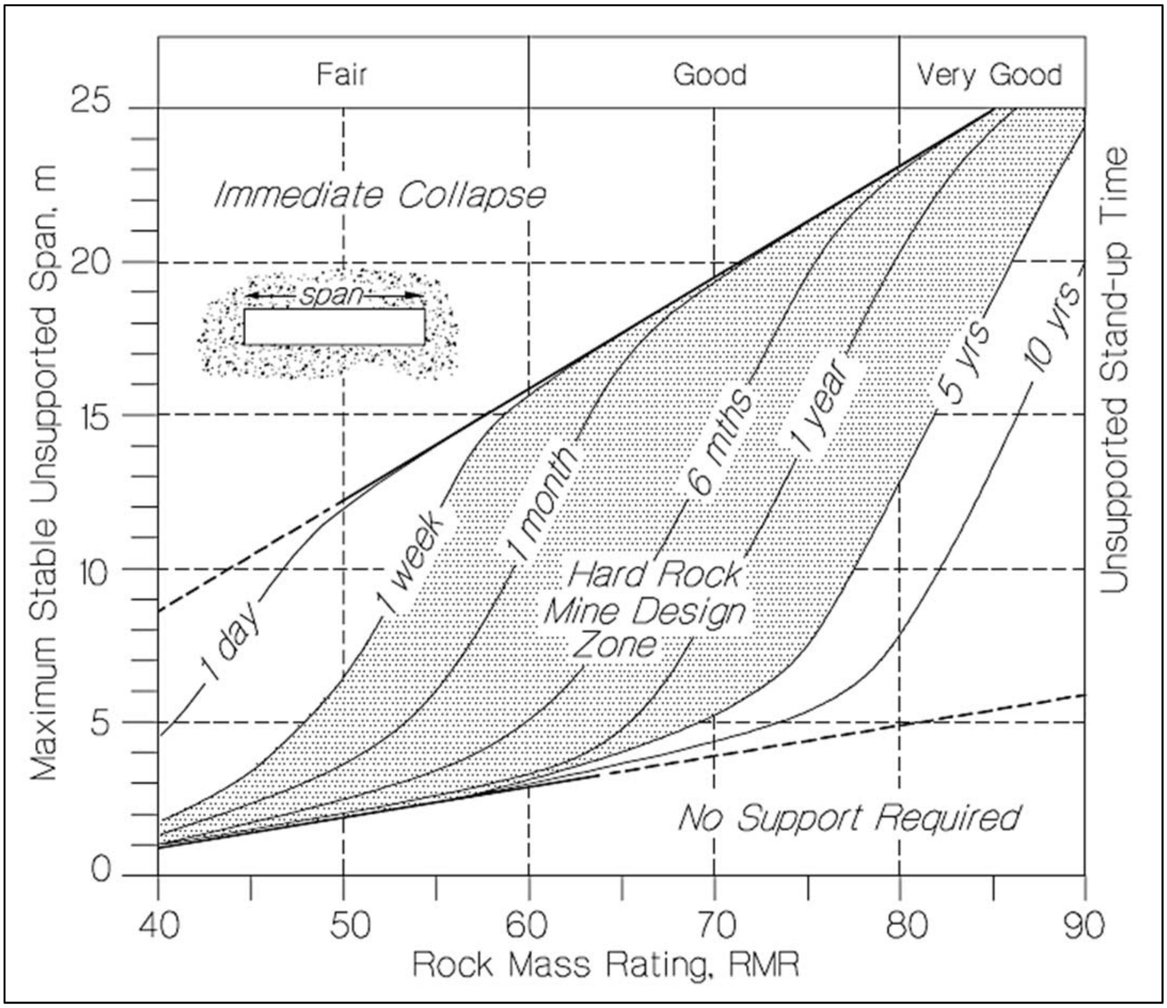


Figure 3-15 Stand-up time guidelines (from Hutchinson and Diederichs, 1996).

3.5.6 **Subjectivity**

Rock mass classification may be subjective, as each person may have different approaches to classifying the rock mass. There is a range of ratings that can be chosen, for example in Table 2-3, and the rating actually selected will depend on what each person observes and how these observations are interpreted. Ameli (2008) compared the results of two separate geological mapping teams and found that there were differences in the rock quality assessment obtained by the two teams.

Some parameters must also be inferred if it is not possible to physically inspect the rock mass. If the exposed rock mass is potentially unstable, direct measurements on the strength of the rock mass or the joint conditions are not feasible. In these cases, previous experience is invaluable for estimating rock mass properties.

3.6 Summary

Assessing rock mass stability and dilution is a complex problem, as shown by the variety and variability of the above parameters. For the scope of this thesis, only the geometry and the geology will be considered, but special note will be made for cases in which one of the above mentioned parameters is suspected to have influenced the amount of dilution.

Excessive dilution has a negative impact on the economic viability of a stope. Much work has been completed to attempt to estimate the amount of dilution that will occur based on the geometry, rock mass classification, stress condition, and various other parameters. Accurate estimation of rock mass properties is essential to achieving a reasonable prediction of the rock mass behaviour. Several other influences on the amount of dilution have been identified, but correlating geology observations to engineering rock mass properties, and standardizing the values for the stope geometries, are the basis of this thesis.

Chapter 4 – Geological Setting and Geology Data Collected for Rock Classification at Eagle Point Mine

This chapter presents the regional geology, details on the unconformity-type uranium deposit, local geology, main rock types, and the common alteration products of the Eagle Point Mine.

Also presented is a summary of the common mapping and core logging techniques used at the mine. Geologists and geological technicians inspect each round to enable mining development to follow the ore; however, they do not specifically collect detailed data for rock classification purposes. As their work occurs with no shielding from the uranium ore, they must spend as little time as possible in the ore headings. Detailed core logging is also conducted to assist with the prediction of the extent of mineralized zones for determining stoping limits, ore tonnage and expected grades. Only limited data is collected for geotechnical purposes due to production and time constraints.

4.1 Regional and Mine Geology

4.1.1 Regional Geology

The Eagle Point Mine is located along the edge of a geological area known as the Athabasca Basin (Figure 4-1). All of the Athabasca Basin uranium deposits are unconformity-type deposits, where Archean and Paleoproterozoic metasedimentary and metaigneous rocks unconformably underlie the Athabasca Basin Proterozoic sandstones (Kotzer and Kyzer, 1995). The Eagle Point Mine deposits are hosted entirely within basement, metasedimentary units of the Paleoproterozoic Wollaston Domain (Belyk, 2007).

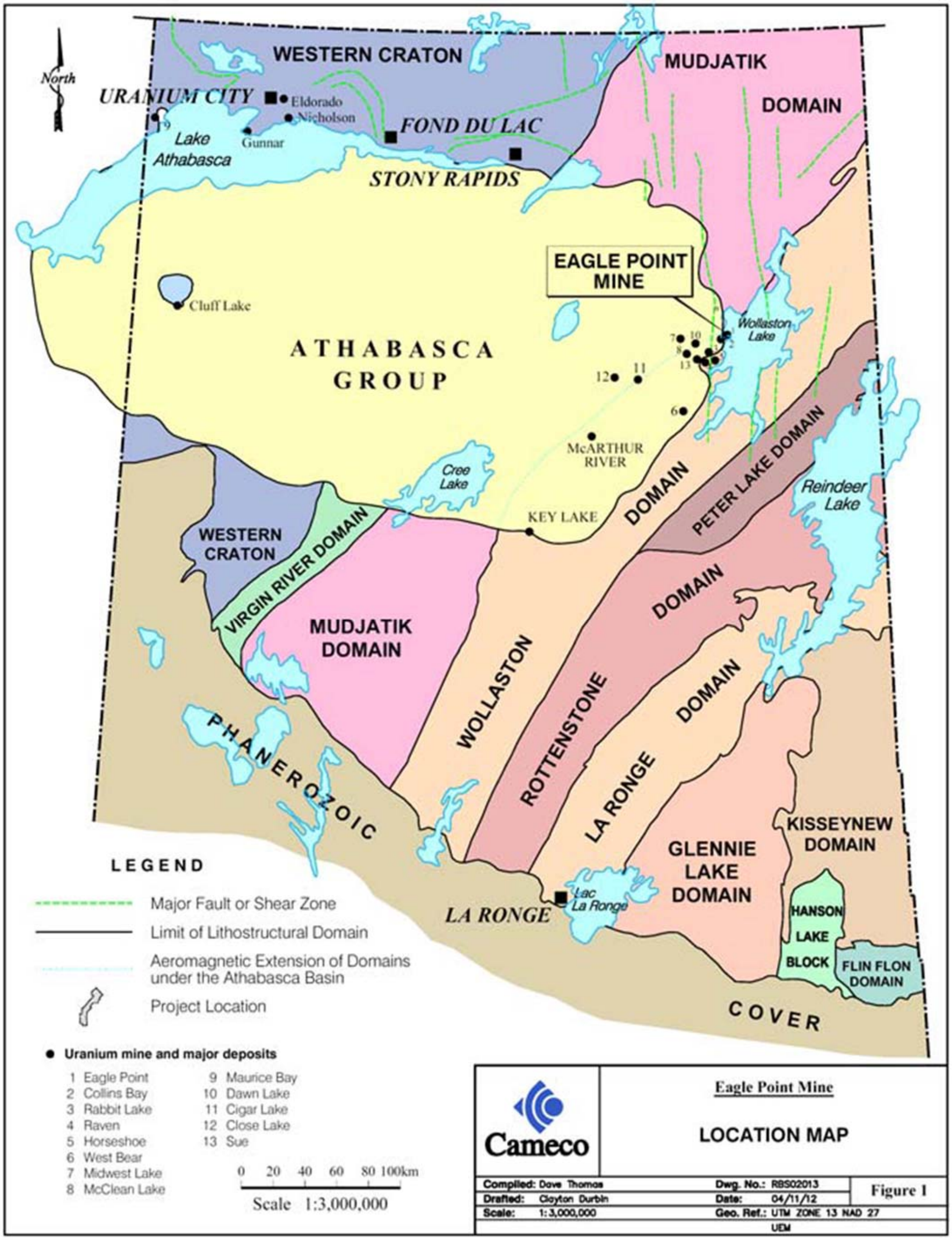


Figure 4-1 Eagle Point Mine location map showing the regional geological groups and domains (From Belyk, 2007).

The dominant structural features of the area are a series of reverse faults and associated fracture zones, to which the uranium mineralization is associated. The Collins Bay fault (Figure 4-2), a regional, north-easterly-trending reverse fault, extends from west of Rabbit Lake, through the Collins Bay B-zone, D-zone and A-zone deposits (Jones, 1980). In the Eagle Point area, the Collins Bay Fault is more or less parallel to the strike and dip of the Wollaston Domain metasediments.

Thomas (2003) discusses the uranium deposition as follows: "The distribution of pitchblende, hematite and bleaching in the replacement style mineralization is consistent with a unidirectional fluid flow utilizing foliation or closely-spaced fracture-controlled permeability with precipitation occurring at localized redox fronts." This fluid flow has significantly influenced the engineering properties of the surrounding rock mass, as well as deposited mineral concentrations which have made mining possible.

4.1.2 **Eagle Point Mine Geology**

Uranium was deposited approximately 1.0 to 1.4 billion years ago (Andrade, 1989) along geological fault zones and fractures in metamorphic rocks of the Canadian Shield. Belyk (2007) states, there are several mineralized deposits within the Rabbit Lake Operation site (Figure 4-2). This thesis is focused on the mineralized zones specific to the Eagle Point Mine as shown in cross-sectional view (Figure 4-3).

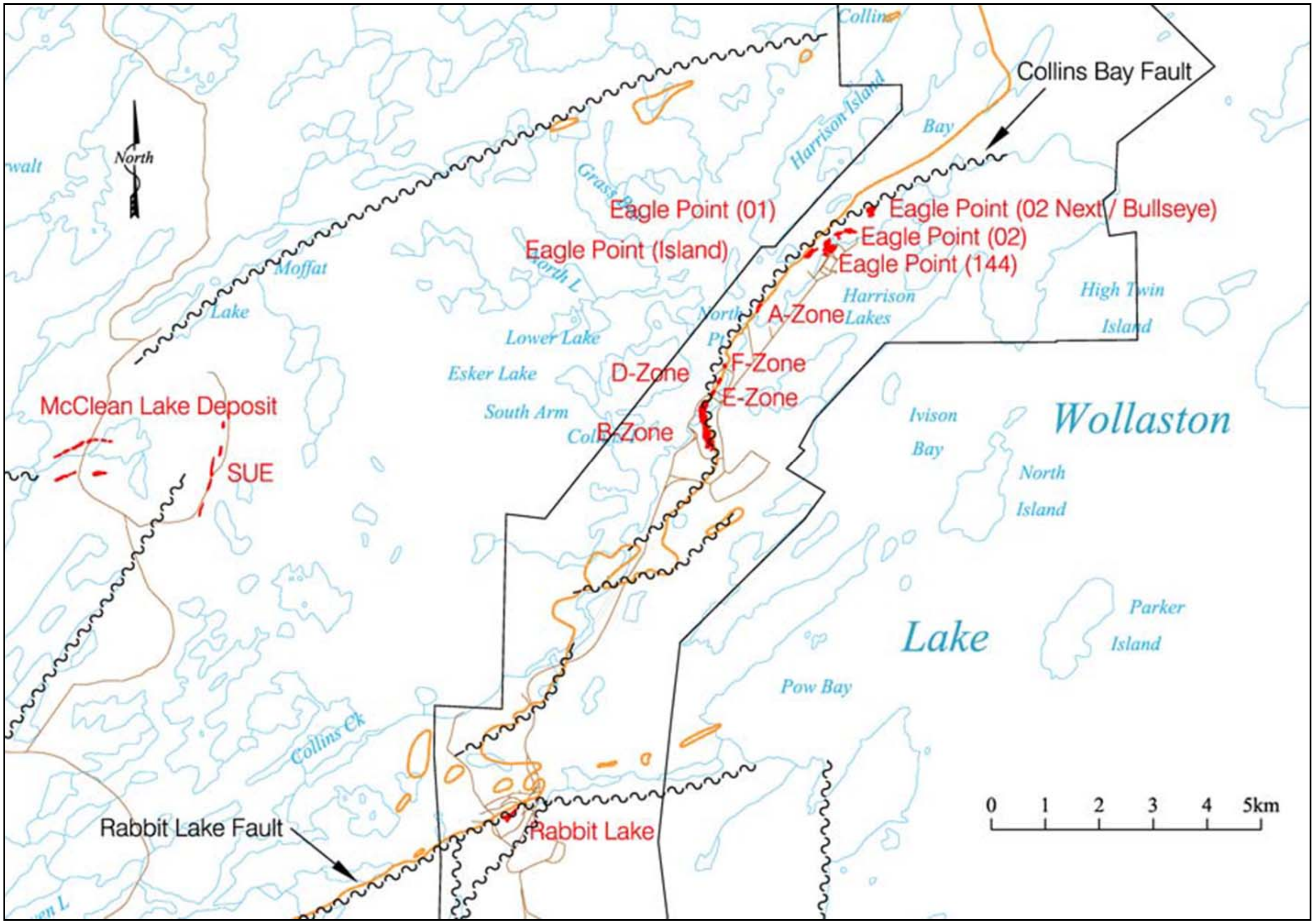


Figure 4-2 Plan view of Rabbit Lake site deposits showing ore bodies within the area of the mine site (From Belyk, 2007). The projections of major fault systems are also shown.

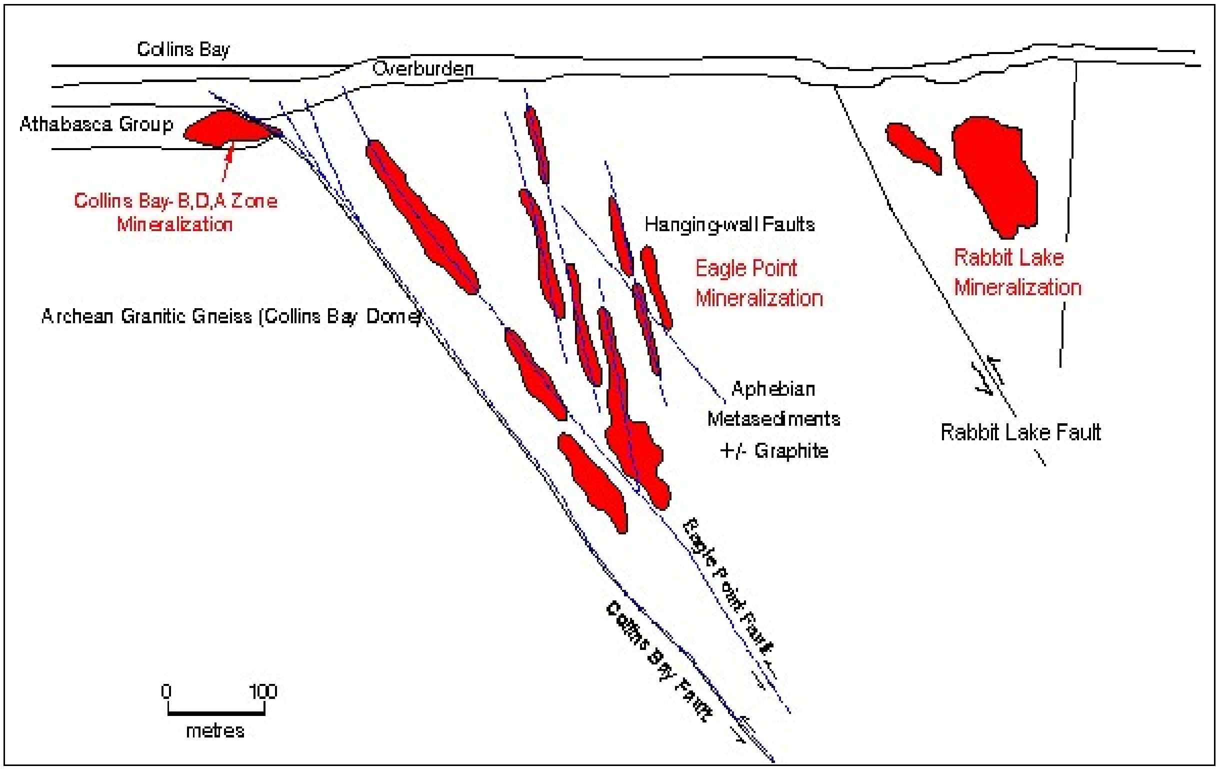


Figure 4-3 Schematic cross-section of uranium deposits at the Rabbit Lake project, looking north (From Dishaw, 2005).

Thomas (2003) subdivided the Wollaston Group into two mine scale sequences (Figure 4-4). The Lower Mine Sequence consists primarily of graphitic pelites and pegmatite. The Upper Mine Sequence is composed of quartzofeldspathic and biotite-quartz-feldspar gneisses. There is a "Transitional Sequence", shown in Figure 4-4, that includes orthoquartzite and rare calc-silicate gneisses which exists between the two main sequences. There is a final group at the lowest point of the stratigraphic column consisting of Archean granodiorites to tonalitic gneisses, also referred to as the Collins Bay Dome. The feldspar porphyry dykes and sills are only recognized in one area of the mine and crosscuts all other lithologies, with the exception of the pegmatite (Figure 4-4).

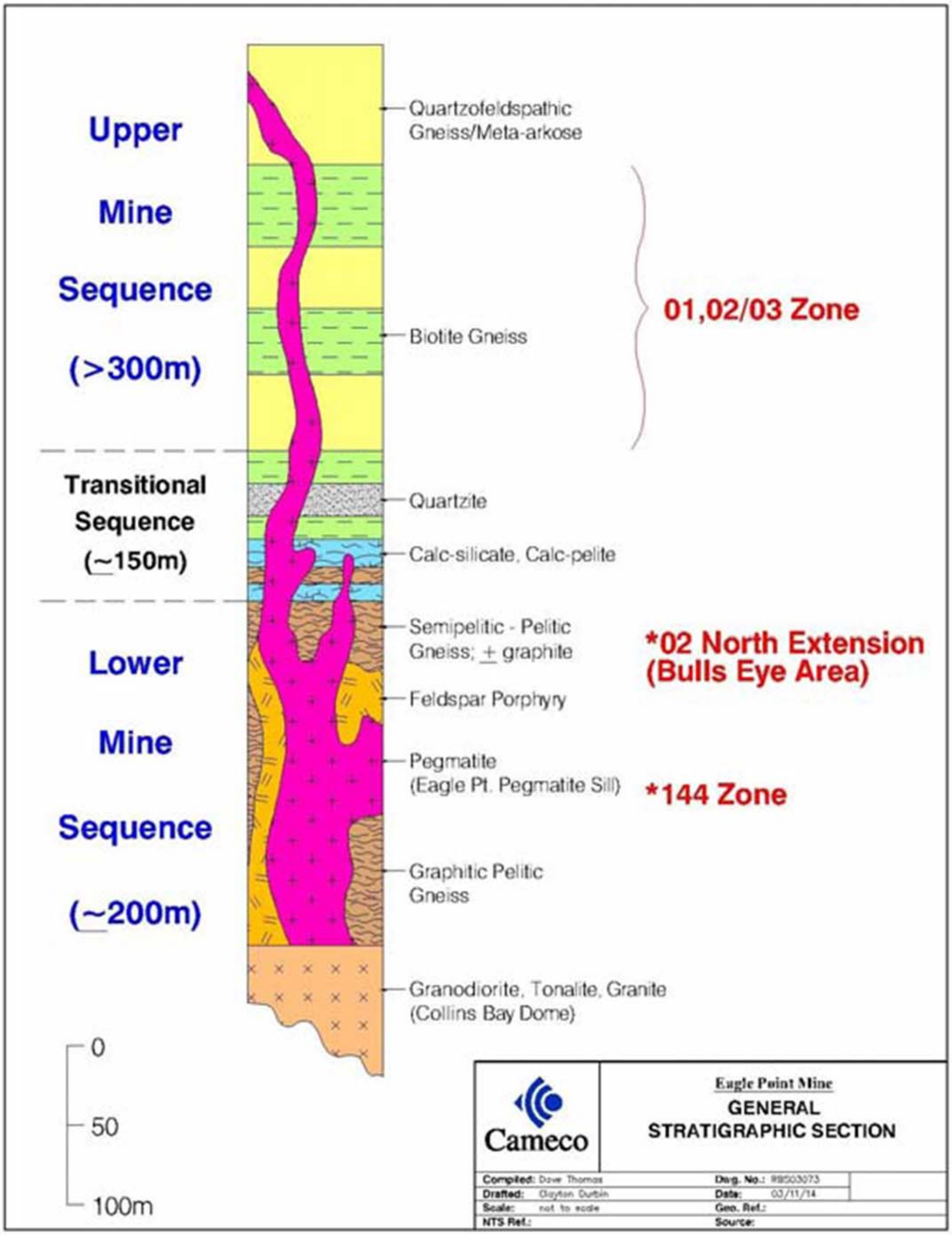


Figure 4-4 Eagle Point Mine general stratigraphic section (From Thomas, 2003).

As the ore mineralization is structurally controlled, it may be found in the transitional sequence and all of the main sequences except for the Archean granodiorites to tonalitic gneisses, or Collins Bay Dome.

Belyk (2007) states, the Wollaston Group metasediments found underground at the Eagle Point Mine are:

1 – Graphitic metapelite gneiss (referred to as Graphitic gneiss) – Contains quartz, feldspar, biotite, and graphite with occasional cordierite and garnet porphyroblasts. Graphite mainly occurs as fine disseminations, foliation controlled concentrations, or as fault/shear bound accumulations. This is found in the Lower Mine Sequence.

2 – Non-Graphitic metapelite gneiss (referred to as Biotite-quartz-feldspar gneiss) – Contains quartz, feldspar, and biotite, with occasional garnet, cordierite, sillimanite and trace pyrite. This is found in the Upper Mine Sequence.

3 – Quartzofeldspathic (referred to as Quartz-feldspar gneiss) – Dominated by quartz and feldspar with lesser amounts of biotite, sillimanite, and garnet.

4 – Quartzite – Primarily quartz with lesser amounts of feldspar, biotite, and sillimanite. It may range from massive to weakly foliated.

5 – Calc-silicate gneiss – Exists within the transitional areas between the Upper and Lower Sequences.

6 – Pegmatite – Granodioritic to tonalitic in composition, with lesser granitic varieties and contain trace amounts of biotite and garnet. Pegmatites are pervasive throughout the entire mine.

7 – Feldspar Porphyry – This is an intrusive geological unit.

4.1.3 **Alteration Products**

Hydrothermal alteration of the rock mass occurred in conjunction with the deposition of the uranium ore. Subsequently, the rock mass near the ore zones is highly altered, and in general, is much weaker than fresh rock located away from the fractured zones. Both pervasive alteration of the intact rock mass and alteration along the joint/fault surfaces have occurred in the fractured and faulted areas (Dishaw, 2005).

Thomas (2003) notes that the alteration types consist of bleaching (a change in colour or change in the crystallinity of micaceous minerals), clay or argillization, pyritization, hematization, carbonate $\pm$ quartz veins, quartz veins, sericitization and chloritization, as shown in Figure 4-5. The alteration may be the replacement of one mineral with another, or the deposition of a mineral. In terms of engineering rock mass properties, these alteration types may be summarized as three groups:

- Group 1 – Softening alteration, including bleaching, clay or argillization
- Group 2 – Low-friction alteration, including sericitization and chloritization
- Group 3 – Other types of alteration, including pyritization, hematization, carbonate $\pm$ quartz veins, and quartz veins.

An example of several types of the alteration products are shown in Figure 4-6. Figure 4-7 illustrates an example of the bleaching, or change in colour, and clay alteration around a graphitic fault structure.

The more fractured the rock mass becomes, the more pathways exist for the hydrothermal fluids to circulate and alter the host minerals. Figure 4-8 shows the progression of alteration of a rock mass from fresh rock to strongly altered and highlights the influence of fractures.

Clay minerals, chlorite, and graphite are widely recognized as being detrimental to the stability of a rock mass. They weaken the rock mass and lower the angle of friction on joint surfaces. These types of alteration are found in the highly altered zones around the ore veins at the Eagle Point Mine.

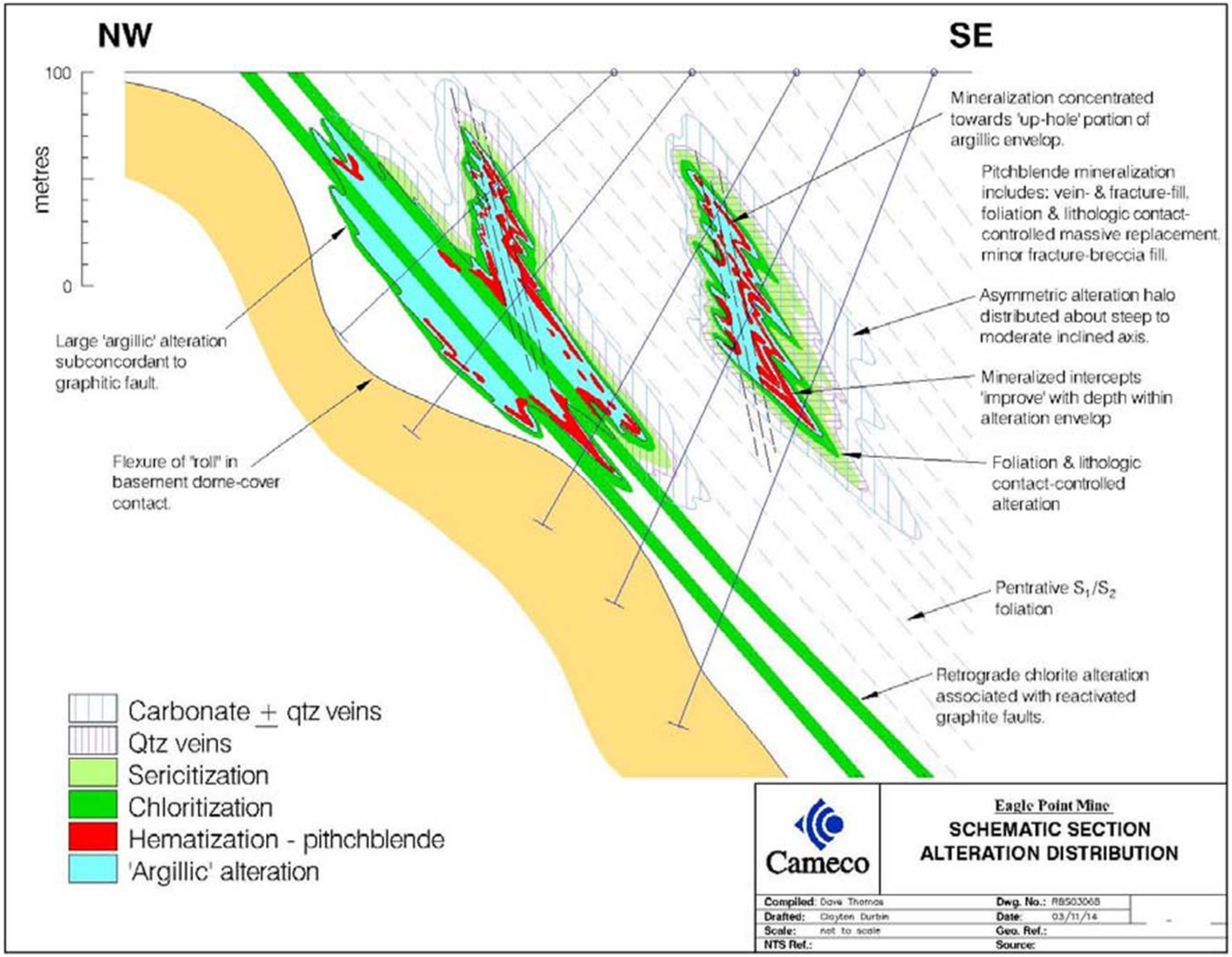


Figure 4-5 Schematic alteration distribution section - Eagle Point Mine area (From Thomas, 2003).

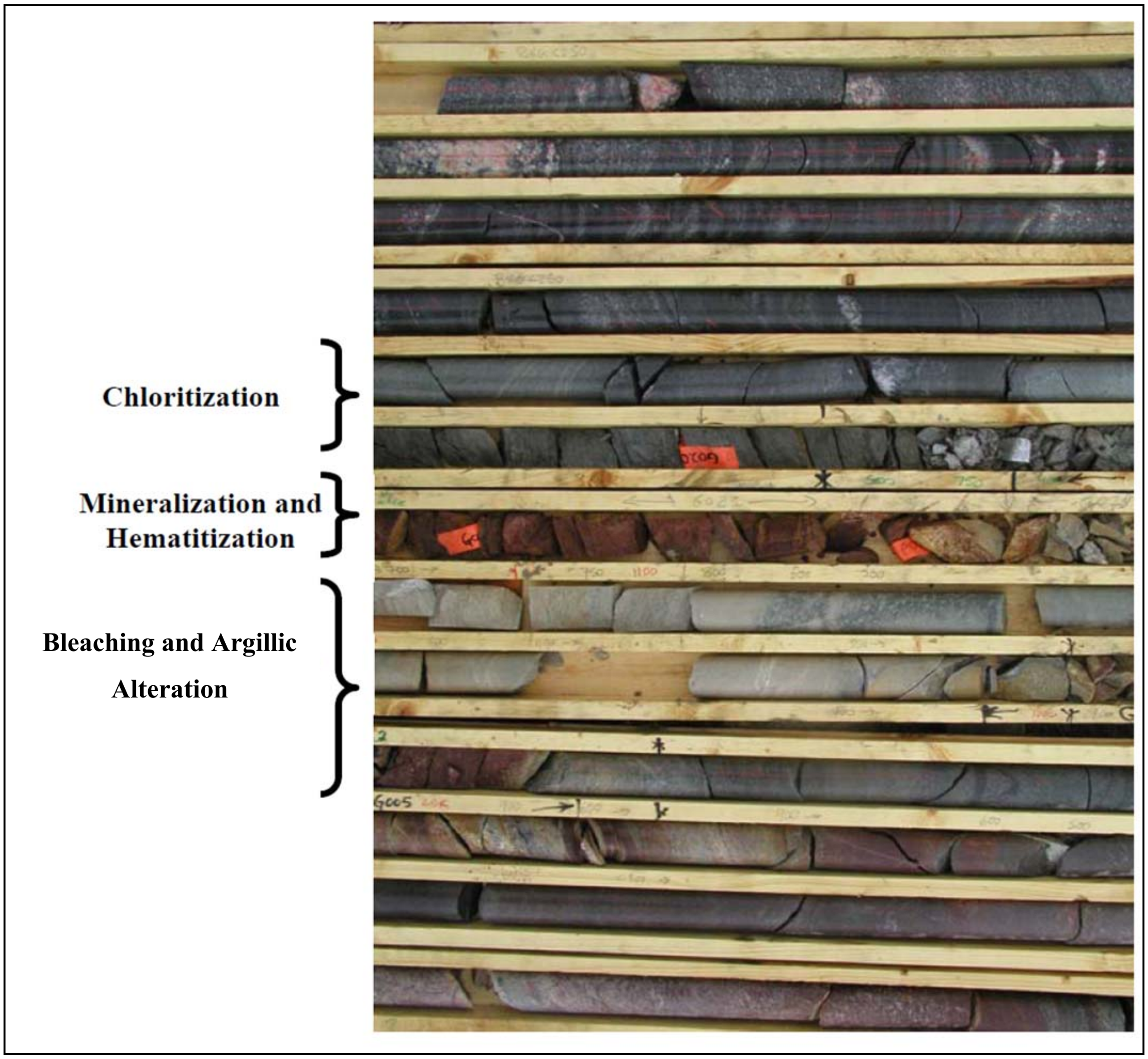


Figure 4-6 Alteration sequence illustrating a transition through a narrow zone of massive dark green chloritization, into a discrete interval of uranium mineralization and hematization, followed by intense bleaching and clay (argillic) alteration in EPE-044, from 226 to 240 meters. (From Thomas, 2003).

Figure 4-7 Intense sericitic alteration and weak to moderate bleaching and clay alteration adjacent to narrow semibrittle graphitic faults in EPE-059, from 277.8 to 286.5 meters. (From Thomas, 2003).

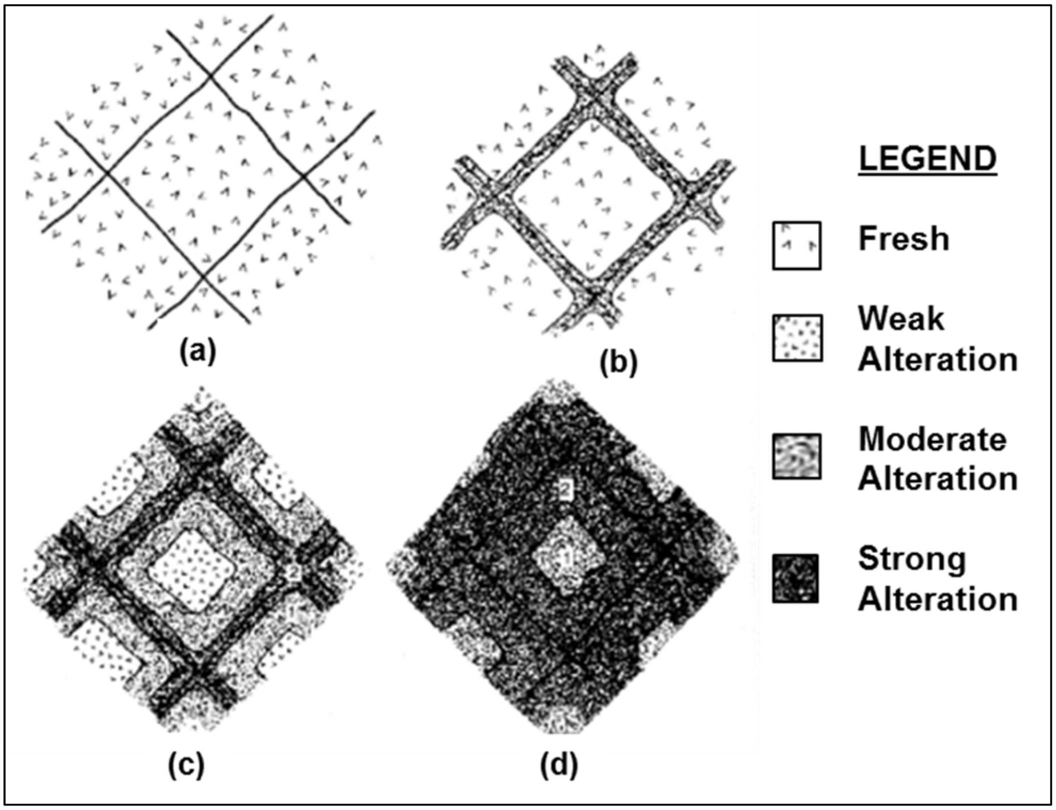


Figure 4-8 Progression of alteration (After Guilbert and Park, 1985). A fresh rock mass is shown in sketch (a), weak alteration is shown in sketch (b), moderate alteration is shown in sketch (c), and strong alteration is shown in sketch (d).

4.2 Geological Approach for Data Collection

4.2.1 **Site Geological Classification System**

The site geology staff developed a site-specific mapping technique to assist in ore body field interpretation. A rock strength and alteration code was developed for describing the rock mass. The Rock Strength assessment follows the Jennings and Robertson (1969) "R" values.

The alteration code ranges from fresh (A1) to weakly (A3) to moderately (A5) to strongly (A7) altered as shown in Table 4-1. This approximately corresponds to Figure 4-8a for fresh rock, Figure 4-8b for weak alteration, Figure 4-8c for moderate alteration and Figure 4-8d for a strongly altered rock mass.

Table 4-1 Underground geological rock mass assessment (conversation, Basnett, R., 1997)

Rock Strength Assessment		Alteration Assessment		
Code	Description	Code	Description	% of rock mass altered
R1	Very Weak Rock Indents with thumbnail	A1	Fresh	0%
R2	Weak Rock Peel with a knife	A3	Weakly Altered	0 – 5%
R3	Medium Strong Rock Several hammer blows to break	A5	Moderately Altered	5 – 25%
		A7	Strongly Altered	25 – 100%

4.3 Sources of Geological Data

The two main sources of geological data collection at the Eagle Point Mine are drift / face mapping and core logging from diamond drill holes. Drift and face mapping are completed for all underground development, so the information is readily available for engineering analysis. However, the information is limited to what can be seen, so there is little information on the rock mass within the hanging wall. Exploration core is also extensive throughout the ore bodies, but the focus of the logging has traditionally been on collecting mineralogical information for the ore body.

4.3.1 **Method of Geological Mapping**

The site Geology Technicians log data for every section of development advance. The walls, back (roof), and face are mapped for structures, lithology, degree of alteration, water inflow, and general shape and size. The mapping of the back is digitized and transferred to the mine design program used by the mine site. This includes the lithology, ore zones, major and minor geological structures, and information on the rock strength and degree of alteration. An example of the field data collected by the site geology department is shown in Figure 4-9.

The radiation produced by the uranium ore limits the amount of time that personnel can spend in an ore drift. This limits the time available for data collection.

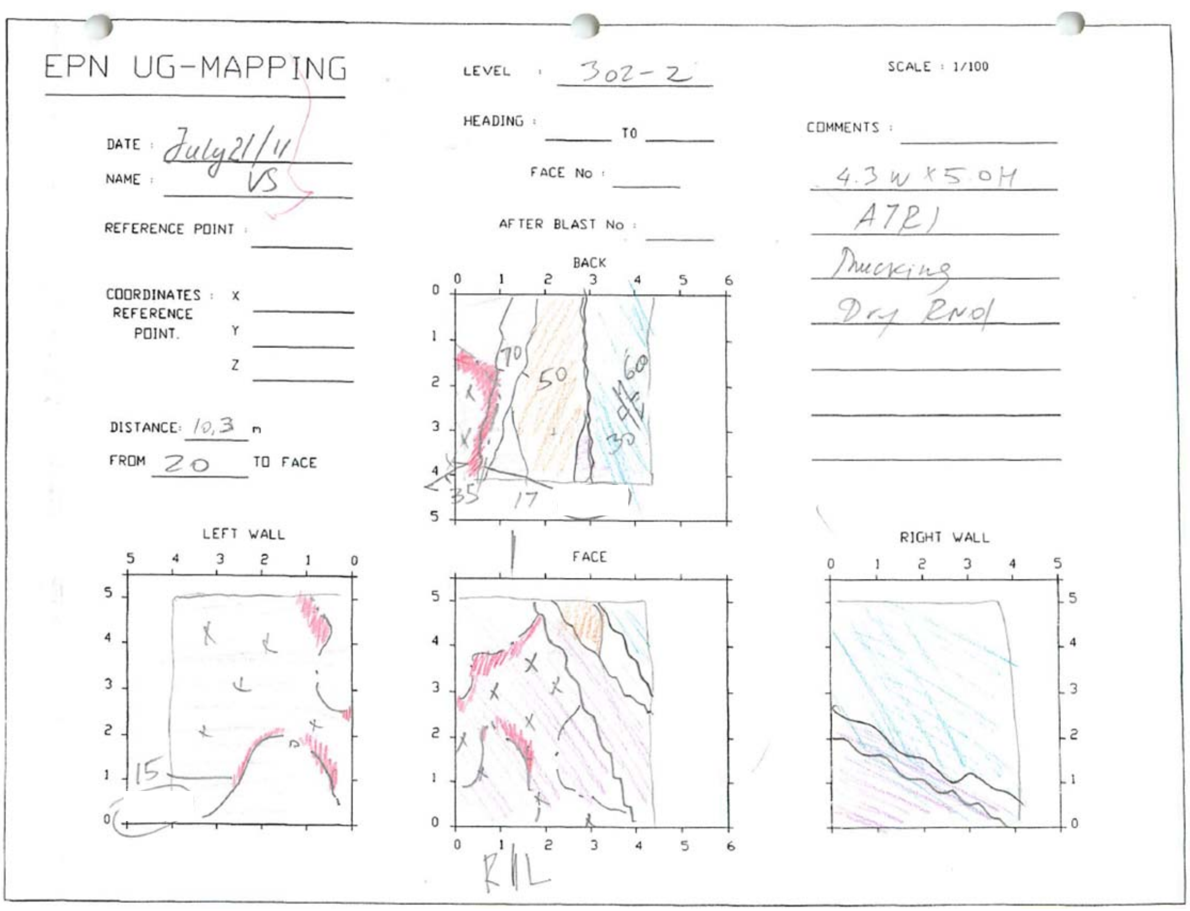


Figure 4-9 Example of mapping completed by the geology department.

4.3.2 **Diamond Drill Hole Core Logging**

Exploration holes are drilled to find new ore zones, delineate known ore zones, provide preliminary grade estimates, and collect geotechnical information. Personnel logging core record information such as, but not limited to, rock type, degree of alteration, alteration mineralogy, fault and shear zones, and RQD. An objective of this project was to develop a simple method of incorporating this data into rock mechanics design.

4.3.3 **Exploration Diamond Drill Hole Core Logging Limitations**

If care is not taken to ensure a variety of drill hole orientations are collected for analysis, there can be a core logging bias introduced to the data. Geological features which dip at angles between parallel, and 15° to the core axis, may not be effectively logged or observed. Jakubec and Esterhuizen (2007) summarized the main challenges in rock mass assessment based solely on core logging as:

- Difficulty in differentiating between natural defects and artificially induced breaks
- Assessment of discontinuities in foliated or highly laminated rocks is more difficult
- Difficulty in differentiating between continuous joints and discontinuous fractures
- Missing or underestimating discontinuity sets subparallel to the drill hole due to drilling orientation bias
- Weak joint infill material being washed out in most drilling processes
- Rock strength assessment in weathered/altered sensitive rock types is more difficult due to the interaction with drilling fluids and disturbance of the sample
- Anisotropy of the material makes assessment of both the intact rock strength and discontinuity strength a problem
- The cross-section of the core is simply too small to capture joint geometry

Figure 4-10 shows an example of a rock mass and the potential fracture features per metre (FF/m) encountered with 2 diamond drill holes at different angles. One drill hole shows 3 FF/m, while the other shows 6 FF/m for the same rock mass. This illustrates the bias introduced by the drill hole core orientation.

Caution should be taken if core logging data is to be used as the sole source of information for geotechnical design. However, when used in conjunction with rock exposures, even with the limitations of the data, this information can be a valuable resource.

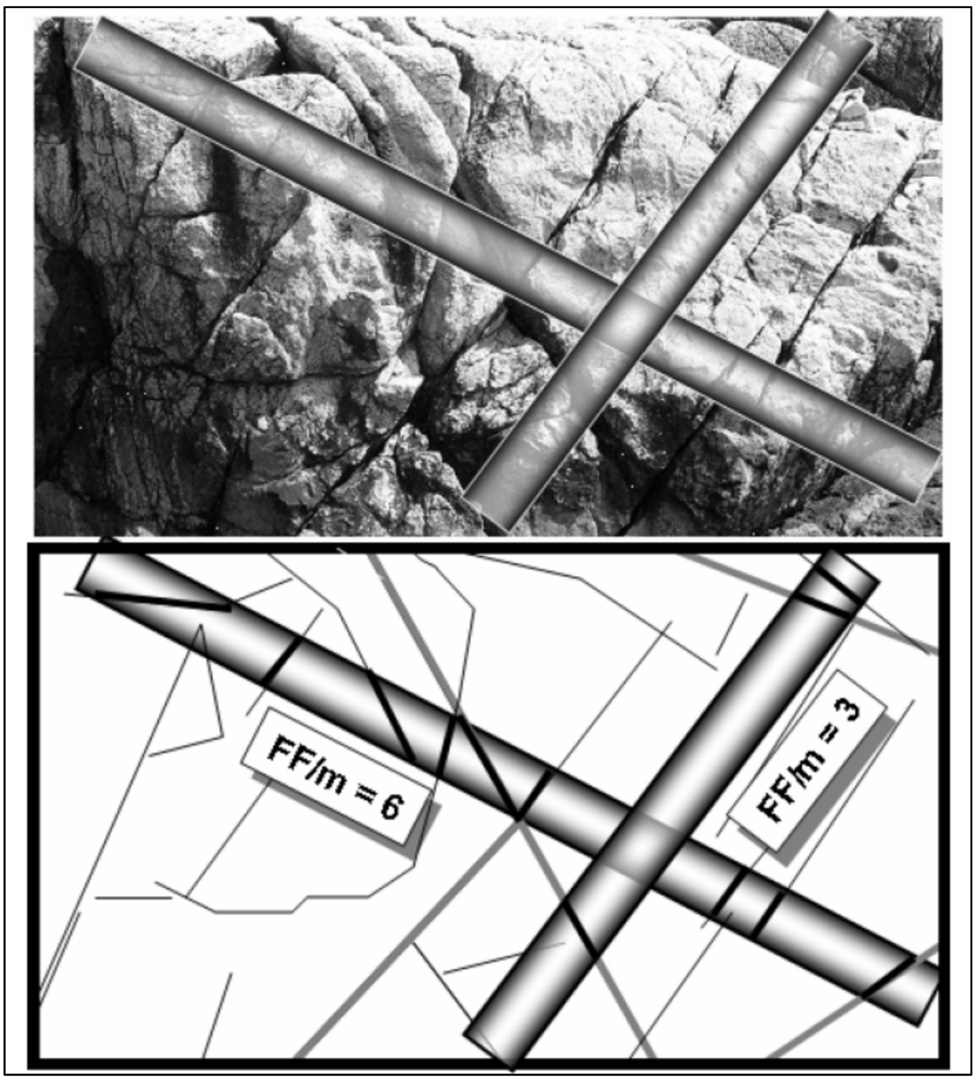


Figure 4-10 Picture illustrating the bias that could be introduced by borehole orientation. Also, it is difficult from the core to judge which discontinuities represent continuous joints and which are small scale fractures (Jakubec and Esterhuizen, 2007).

4.4 Summary

The Eagle Point deposit is found within the Athabasca Basin geological domain. There are several zones of mineralization within the mine, located within Wollaston Group rock types. Due to the mineralization deposition, there are several types of alteration which occur in conjunction with the uranium ore. The Geology staff use an assessment of the alteration to help locate and predict trends in the location of mineralization. Engineers may use the alteration descriptions to predict joint properties and estimate the rock intact strength. The following chapter discusses the application of the geological characterization of the rock mass and the application of this data for engineering.

Chapter 5 - Rock Mass Rating Classification Values Compared to Sources of Available Data

This chapter introduces the site-specific geological mapping techniques used at the Eagle Point Mine, including geological mapping, geotechnical mapping, and exploration diamond drill hole core logging. The advantages and limitations of each method are discussed. The available data, including alteration/rock strength assessments, point load testing of exploration core, RMR_{76} results from audits completed by the site's external Rock Mechanics consultant, and site personnel RMR_{89} ratings, are introduced. The general modes of hanging wall failure are discussed to highlight the rock mass properties which control stability.

5.1 Modes of Stope Surface Failure

Two of the failure modes that may occur for stope hanging walls are block failure or ravelling failure (Figure 5-1). For block failure, the strength and orientation of the joints and or shear zones in the rock mass will have the largest influence on the shape of the failure. Block size is also a factor, however, as long as potential block failures are within the stope limits and inside the stope zone of relaxation, block failure may occur. When the rock mass block size is relatively small compared to the opening surface, a raveling failure can occur. Ravelling failure at the Eagle Point Mine is related to highly foliated gneissic rocks. In the more massive rock types, such as the pegmatite, both failure modes may be observed. In both failure modes, the condition of the joint surfaces will have an effect on the final failure shape.

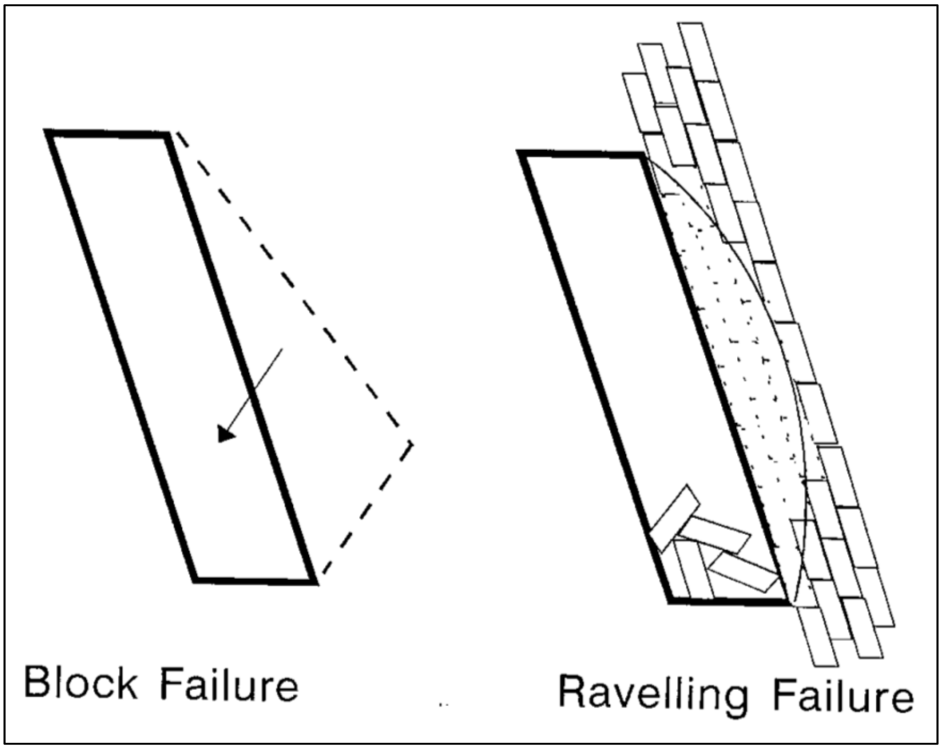


Figure 5-1 Examples of a hanging wall block failure and ravelling failure (From Milne, 1997).

5.2 Comparison of Data Gathering Methods used at Eagle Point Mine

There are three main methods of gathering geotechnical and/or geological information at the Eagle Point Mine. The first, geotechnical mapping, is data collection of a rock exposure with particular attention to the engineering properties of the rock mass. ASTM D4879 - 08 Standard Guide for Geotechnical Mapping of Large Underground Openings in Rock (ASTM, 2008) states that one component of geotechnical mapping is an emphasis placed on those geotechnical features which are anticipated or are found to affect overall performance of the excavation. Geological mapping is the second method, and is data collection of a rock exposure for rock type, mineralogy, structure, alteration, and other geological features of interest. The third method is diamond drill hole core logging, and both geotechnical and geological information may be gathered, but the size of the core may limit the amount of visible information. The following sections describe the advantages and limitations of each of these methods.

There are some common limitations for both geological and geotechnical mapping methods. As shown in Figure 3-15 for the Stand-Up Time Guidelines, the weaker the rock mass, the shorter the length of time that the unsupported span will remain stable at a given size. The installation of primary ground support, in a timely manner, is critical to the stability of an opening. For the Eagle Point Mine, shotcrete application for gamma ray shielding is also required as soon as reasonably possible to protect the underground staff from radiation sources. The combination of generally poor ground conditions and radioactive mineralization dictates that all production tunnels are shotcreted almost immediately after removal of the rock blasted during development. There is a very small window of time where the rock surfaces are exposed. The waste rock areas may be of sufficient quality as to not require shotcrete, but these areas are not usually indicative of stope hanging wall zones.

A second consideration is that it is commonly prohibited in mines to enter an area of unsupported ground. Certain rock mass properties may be observed without direct physical inspection of the rock mass, but qualities such as joint roughness and infilling are difficult to ascertain when the rock mass cannot be physically examined.

5.2.1 **Geological Drift Mapping**

Geological drift mapping is conducted by geology staff to collect data which may influence the location and continuity of the ore zone. Data collected concentrates on major discontinuities which may offset the ore, and the degree of alteration which often indicates the likelihood of ore. The advantage to this method is that geological drift mapping is completed on every section of tunnel advance underground, and the data is easily available and extensive. One shortcoming to the mapping method, specifically for the A/R classification system, is the subjectivity of the categorization of the rock mass. In particular the alteration category, A, is highly dependent on the observations and experience of the geologist collecting the data.

At times an overall alteration value may be assigned to an area, or more specific zones of alteration within an area may be noted. The extremes of the ranges, A1 and A7 for alteration, and R1 and R3 for strength, are more likely to be consistent across all personnel collecting the data. It is the mid ranges, for the alteration category specifically, which will likely be the most variable between the observers (Pagnin, 2009).

Geologists may be able to collect geotechnical information at the same time as the geological information, however, this has not been recommended at the mine site. Radiation dosage is a combination of the intensity of the radiation and the length of time a worker is exposed, and is to be kept as low as reasonably achievable. The additional time collecting detailed information may be considered a health and safety risk.

The geological drift mapping provides the most detailed information on the rock type and major structures; even joint orientations and the number of joint sets may be interpreted from mapping sketches. The greatest advantage is that all areas are mapped geologically to provide direction for drift development to follow the ore.

5.2.2 **Geotechnical Area Mapping**

Geotechnical area mapping is the most desirable source of rock mass classification information, but there are limitations to this method, and the data which are collected.

A limitation to geotechnical area mapping is that even if the rock mass may be mapped for rock mass classification values during development, the exposed area may only be indicative of the rock mass classification of the immediate ore zone, and would have limited information for the immediate hanging wall zone. As the mineralization at the Eagle Point Mine is structurally controlled, the underground drifts that intersect the uranium ore tend to undulate and follow the geological structures. Some stope areas may then have several meters of exposed hanging wall rock, while others may have none at all. Area mapping, when possible, may provide the best information on general properties for a zone such as number of joint sets and potentially joint surface properties.

5.2.3 **Diamond Drill Hole Core Logging**

Exploration diamond drill hole core is often the first and most detailed source of information at the beginning stages of a mining project. Throughout the life of a mine, core can be logged for delineation and resource estimation.

One drawback to using core logging information is that the accuracy of detailed information and the level of detail may be sacrificed in order to keep up with the volume of core to be logged. The primary goal of most exploration core logging is resource delineation, and geotechnical information is seen as a secondary benefit.

Information which is routinely collected that may be used for geotechnical classification is rock type, alteration, the presence of major structures, core recovery, and RQD. The limitations of scale dictate that the collection of information regarding overall joint roughness and number of joint sets is less accurate.

Many of the underground exploration diamond drill holes occur approximately normal to the stope hanging wall surfaces. The RQD values from these holes are very useful and indicative of the immediate hanging wall conditions. There is, however, a bias introduced by using diamond drill hole core as illustrated in Figure 4-10, and it should be considered when using diamond drill hole RQD information.

Caution should be exercised when using diamond drill hole information for geotechnical design, as a rating for a parameter from diamond drill hole information is analogous to a single point measurement on a large surface. For homogenous rock masses, this is less of a concern, but for highly variable, complex geologies, each data point will have less influence on the rating of the surrounding area than the value that could be obtained from drift mapping.

5.3 Exploration Core Data

Exploration diamond drill hole core can provide a lot of valuable information, including lithology, alteration, RQD, core recovery, rock strength, and joint infilling. Generally, the focus for logging is on geological information, but many engineering parameters can be estimated or measured as well. The following section discusses a study completed for this project to attempt to relate the rock strength to the rock type and degree of alteration.

5.3.1 **Point Load Testing on Exploration Core**

Recent research has related dilution to different properties of the rock mass. For example, Capes et al. (2005) found a link between the RQD of the rock mass from exploration core and dilution measured at the George Fisher Mine in Australia, and were able to use it to predict the stability of stope hanging walls. Similarly, a study was completed for this thesis by the author using Eagle Point data to attempt to correlate point load estimates of exploration core strength to the degree and type of alteration (Forster et al., 2006). It was hoped that rock strength would be a measure of alteration, and that this would remove user subjectivity from the assessment. For this research, point load strengths were converted to UCS values based on the International Society of Rock Mechanics (ISRM, 1985) Commission on Testing Methods, Suggested Method for Determining Point Load Strength. The standard point load test diameter is 50 mm, but the uncorrected point load strength index (I_s) may be calculated from any sample, as follows:

$$I_s = \frac{L \times A_e}{D_e^2}$$

(Equation 5.1)

where,

I_s = point load strength index in MPa

L = reading of maximum pressure in the jack piston in MPa

A_e = effective area of the jack piston in square metres (m^2)

D_e = equivalent core diameter in metres

The corrected point load strength index value, or $I_{s(50)}$, may be calculated using the following formula.

$$I_{s(50)} = \left({D_e}/{50} \right)^{0.45} \times I_s$$

(Equation 5.2)

where,

$I_{s(50)}$ = corrected point load strength index in MPa corresponding to a core diameter of 50 mm

D_e = equivalent core diameter in mm

I_s = point load strength index in MPa

Bieniawski (1975) and Broch and Franklin (1972) concluded that the relationship between the UCS and the point load strength could be expressed in the following formula.

$$UCS = 24 \times I_{s(50)}$$

(Equation 5.3)

where,

UCS = unconfined compressive strength in MPa

$I_{s(50)}$ = corrected point load strength index in MPa

Comparisons between the estimated UCS and the degree of alteration for each of five common rock types are shown in Figure 5-2 to Figure 5-6. Geology staff logging diamond drill hole core state the percentage of each mineral alteration, but do not use the same A/R categories as the underground mine geologists. These percentages, as stated in the exploration core logs, were summed by the author to determine an overall alteration number for each point load test. The ranges of alteration for the purposes of analysis correspond to the alteration ranges stated in Table 4-1. Histograms showing the average strength corresponding to the fresh, weak, moderate and strong ranges of alteration are shown with the data. Sample points which were unusually low, or where it was noted that failure corresponded to pre-existing weakness planes along a foliation or fracture surface, were removed from the data set shown. The results of the point load testing may be found in Appendix A.

Table 5-1 was created to compare the effect of alteration on the estimated UCS for rock types. The alteration was defined as fresh (no alteration), weak (0 to 5% alteration), moderate (5 to 25% alteration), and strong (greater than 25% alteration). The trend from the graphical presentations (Figure 5-2 to Figure 5-6) and Table 5-1 is that high strength values correspond to low alteration values, however, low strength observations were found for all degrees of alteration.

Table 5-1 Average point load test estimate of UCS of exploration core by rock type and alteration zone

Rock Type	Degree of Alteration			
	Fresh (0%)	**Weak (0 to 5%)**	**Moderate (5 to 25%)**	**Strong (>25%)**
Graphitic Gneiss	No info	No info	112 MPa	92 MPa
Biotite-Quartz-Feldspar Gneiss	235 MPa	274 MPa *(only one measurement)*	111 MPa	67 MPa
Quartz-Feldspar Gneiss	No info	No info	178 MPa	88 MPa
Pegmatite	No info	No info	86 MPa	74 MPa
Feldspar Porphyry	190 MPa	185 MPa	128 MPa	78 MPa

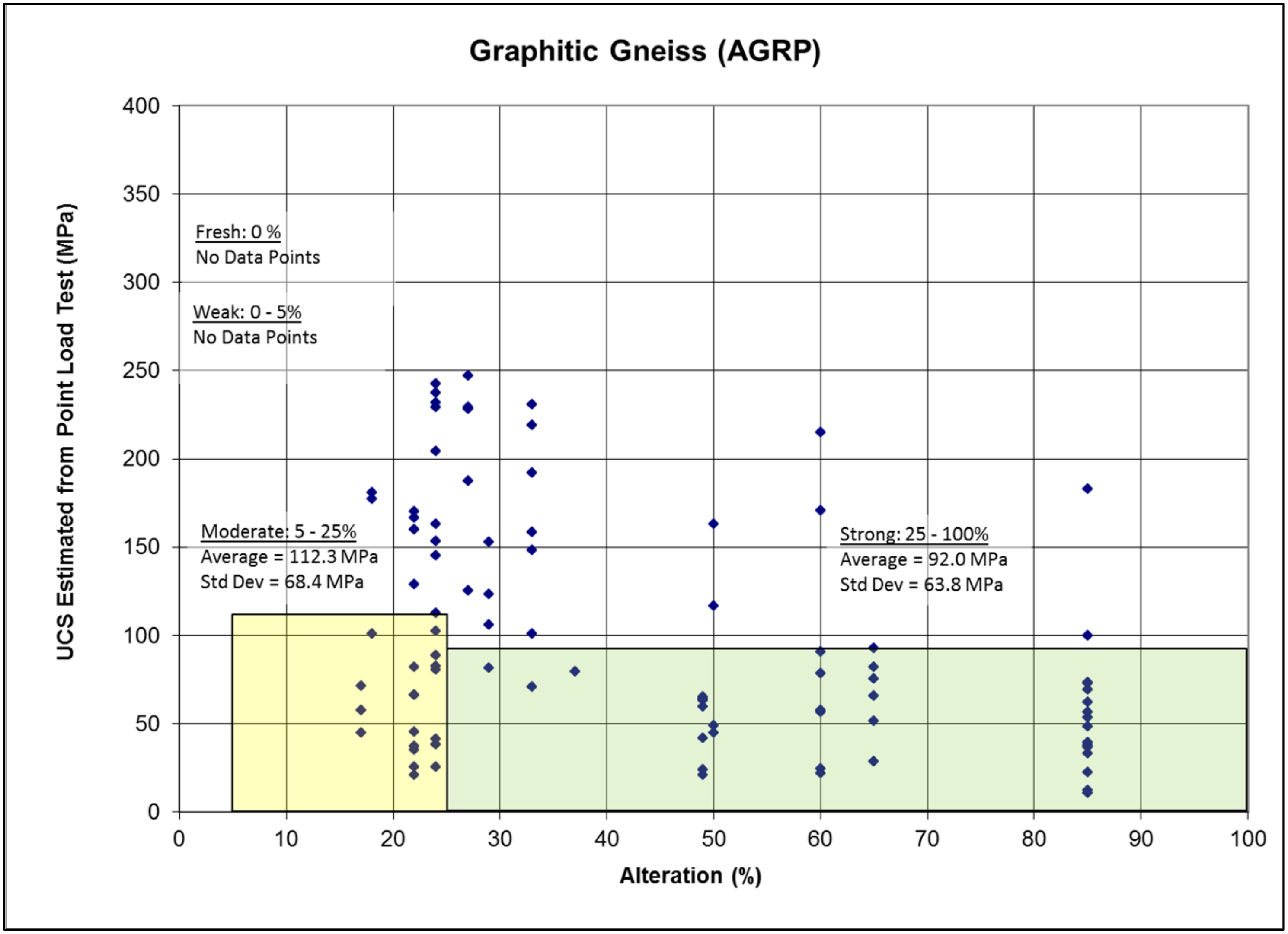


Figure 5-2 Point load test estimate of UCS of exploration core compared to the percentage of alteration for graphitic gneiss samples.

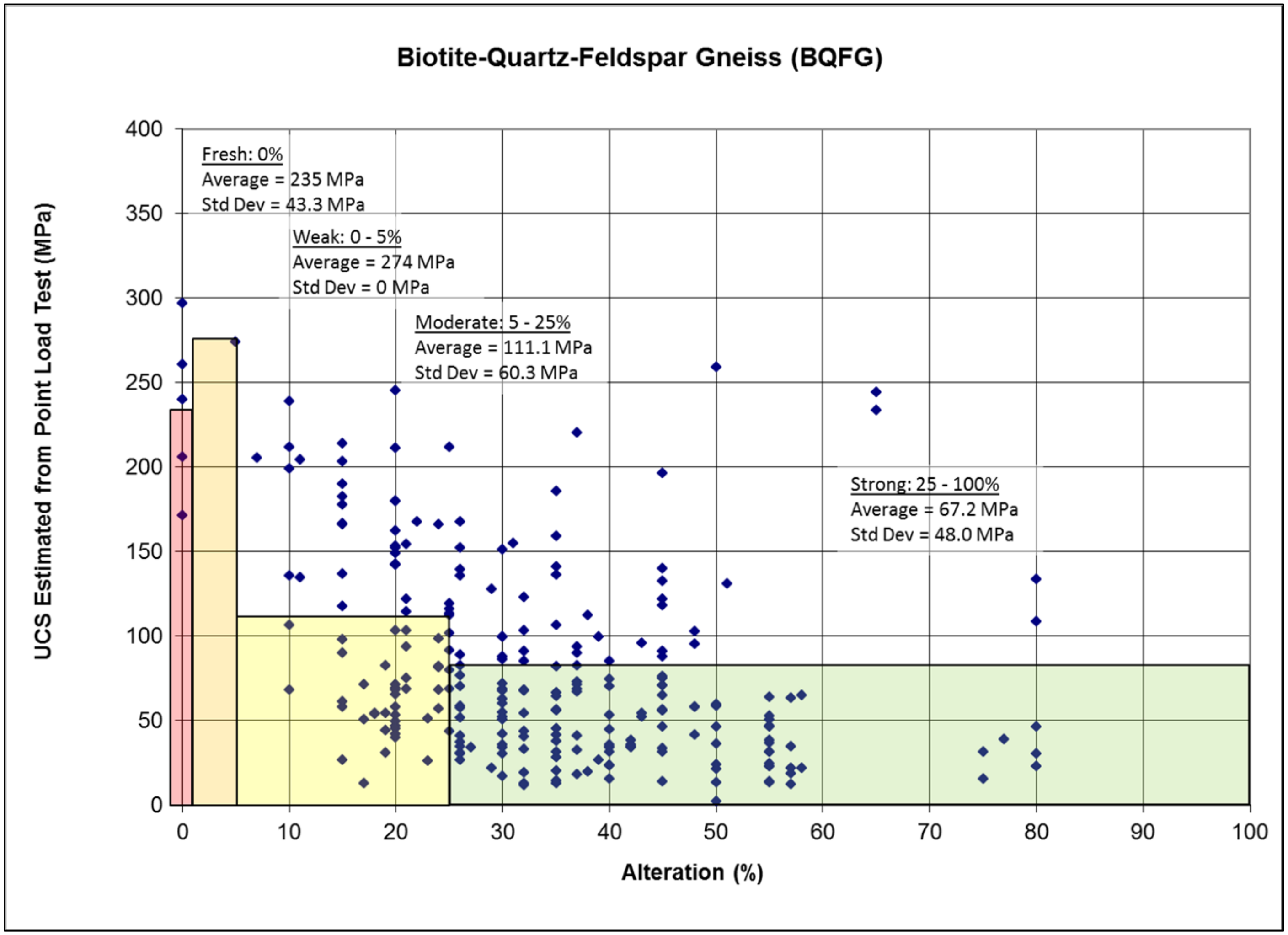


Figure 5-3 Point load test estimate of UCS of exploration core compared to the percentage of alteration for biotite-quartz-feldspar gneiss samples.

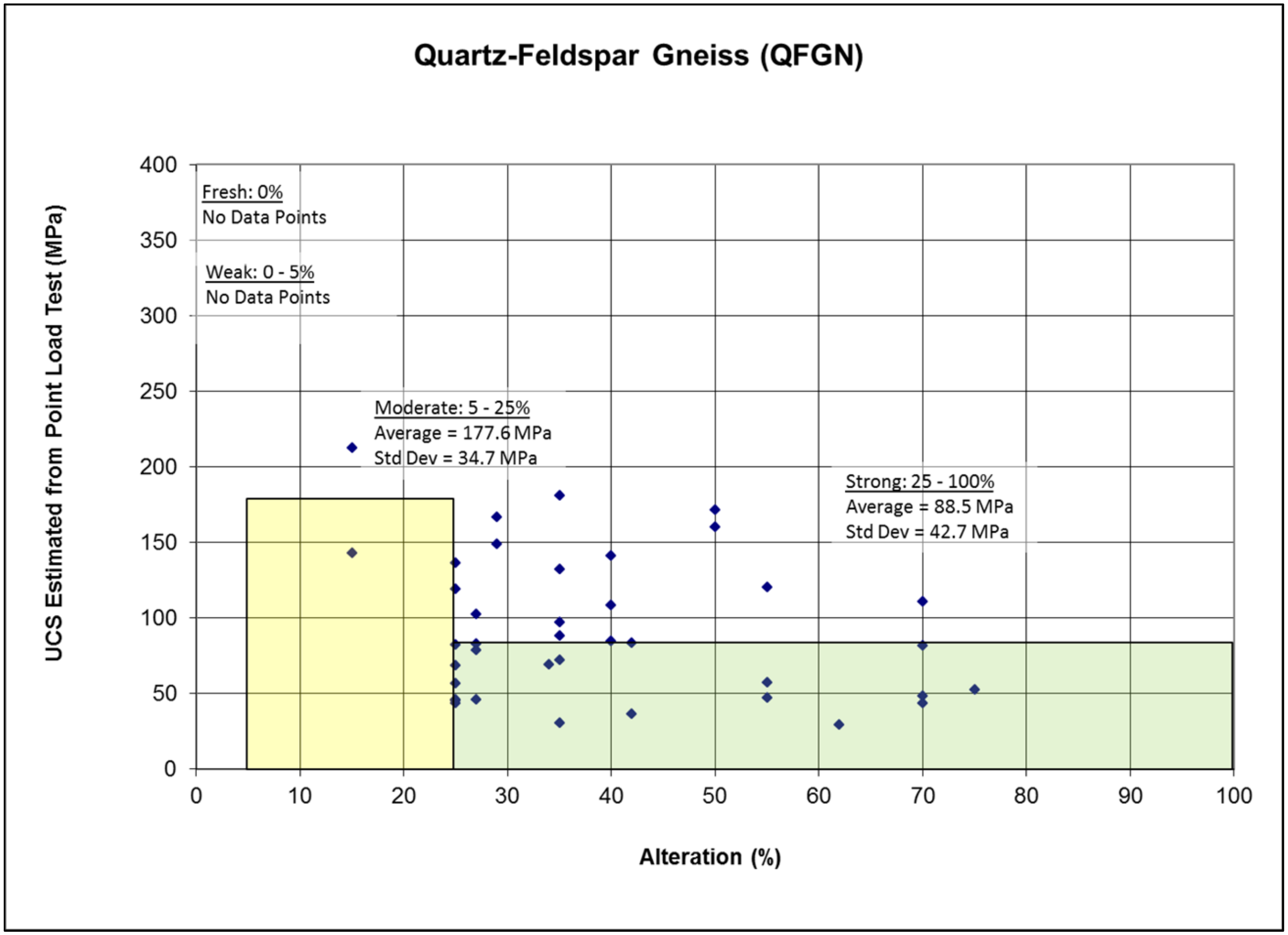


Figure 5-4 Point load test estimate of UCS of exploration core compared to the percentage of alteration for quartz-feldspar gneiss samples.

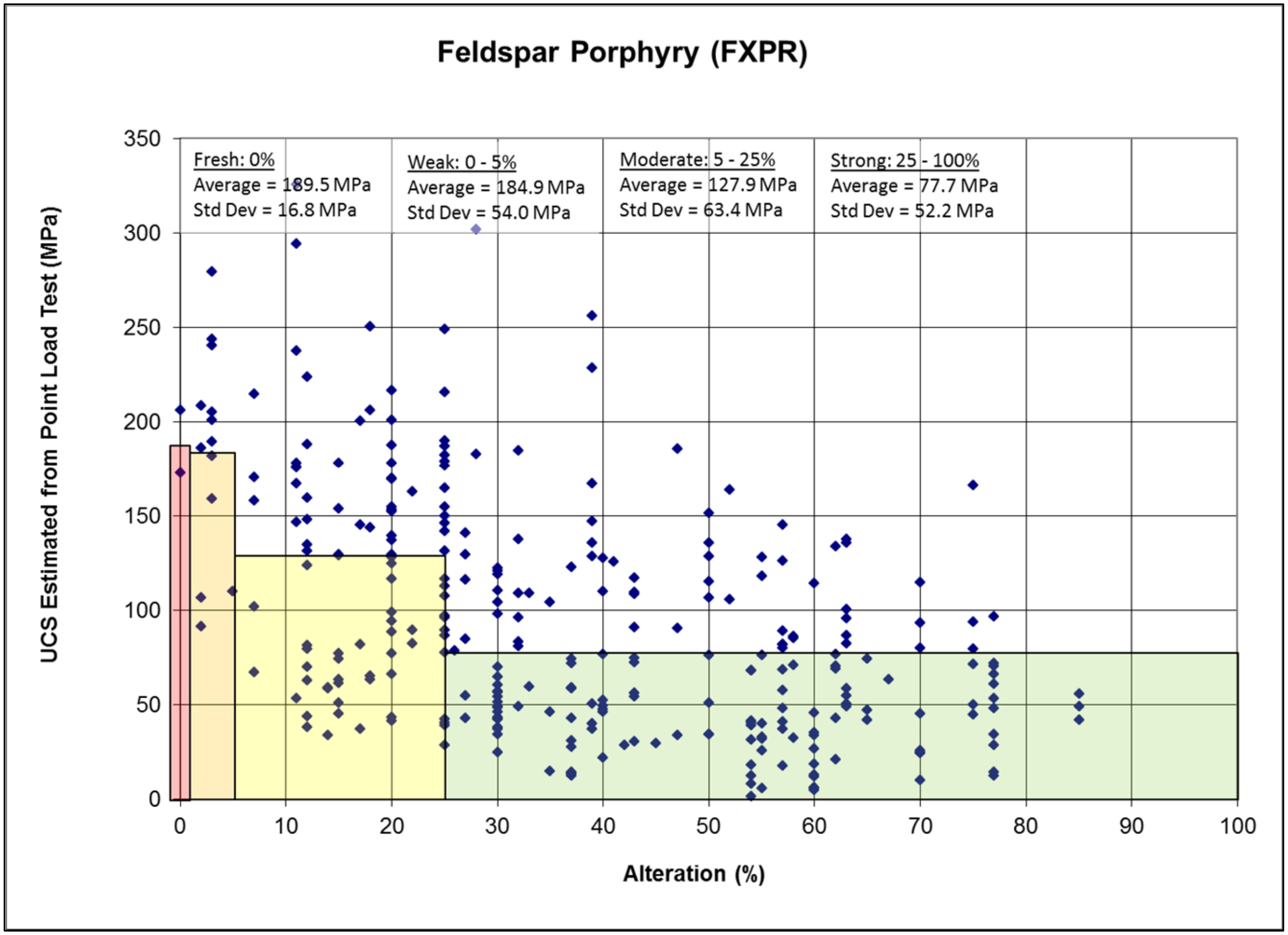


Figure 5-5 Point load test estimate of UCS of exploration core compared to the percentage of alteration for feldspar porphyry samples.

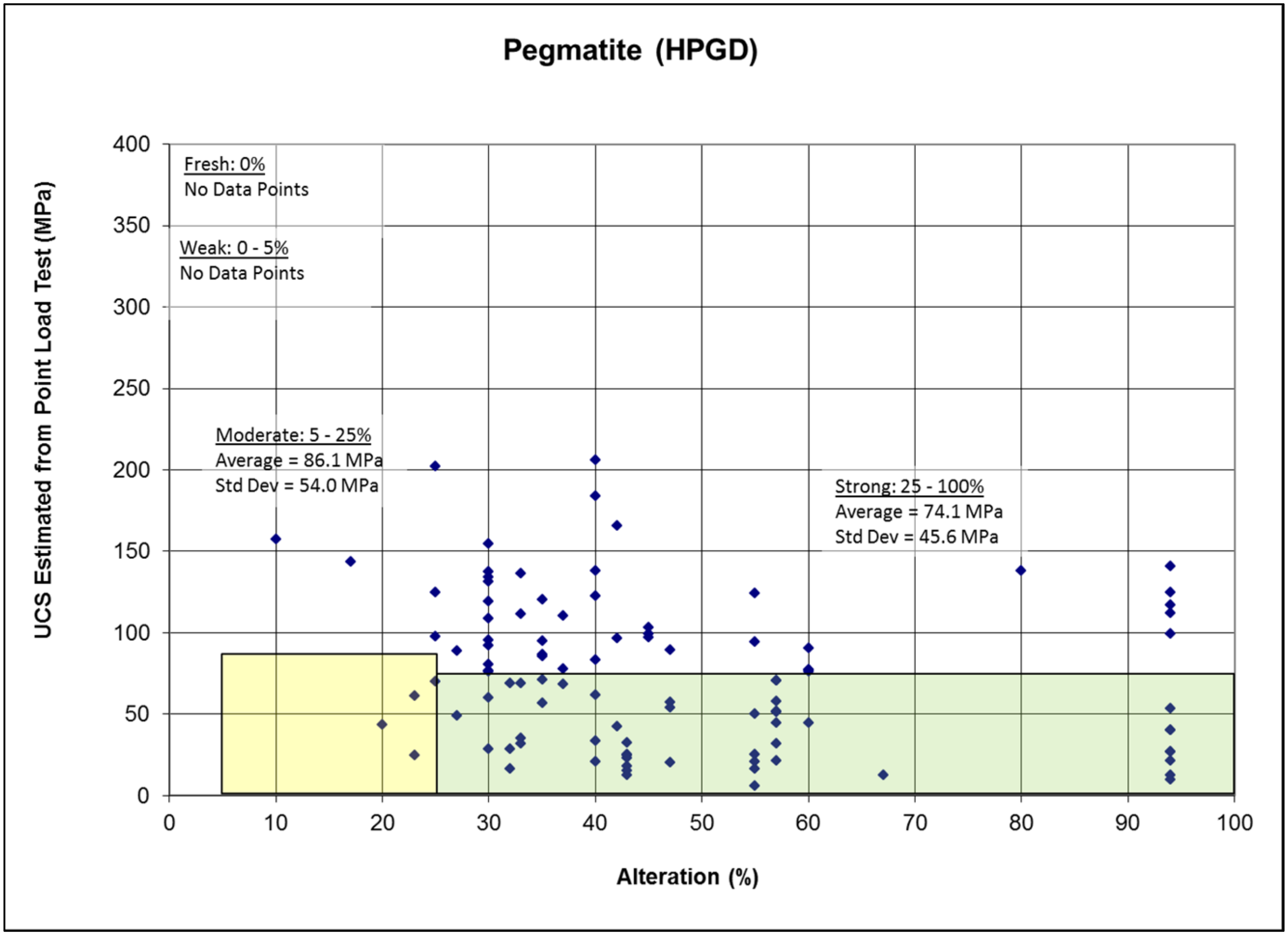


Figure 5-6 Point load test estimate of UCS of exploration core compared to the percentage of alteration for pegmatite samples.

5.4 Geological and Geotechnical Mapping Data

There are several available sources of information which may be used to link the geological mapping observations with rock mass classification values. The geology staff at the mine have procedures in place to map each section of development advance, so rock type and A/R observations are extensive. A previous study was completed to correlate a Q rating for each A/R combination, but it was based upon limited data. There are many historical RMR_{76} ratings which have been completed by an external rock mechanics consultant. There are also RMR_{89} ratings completed by the Mine Engineering staff. The following sections will present each of these sources of data and discuss how each may potentially be used to provide a correlation between the geological A/R value and a Q' value which may be used for stability design and dilution prediction.

5.4.1 **Sutton (1998) Comparison Table of Geological Mapping and Q'**

A study was conducted by Sutton (1998) to link stope stability to the alteration/rock strength assessment that was developed and currently used by geology staff. Available exposures were classified by the alteration and strength categories that were introduced in Table 4-1, and were also mapped for rock mechanics classification purposes.

From this study, nine combinations of alteration (A) and rock strength (R) assessments were identified (Table 5-2, Column 1). Rock masses with both low strength and fresh to weak alteration (A1/R1, A1/R2, and A3/R1) are not found on site and are not shown in Table 5-2. Of these nine categories, six of the A/R combinations were found and mapped for rock mechanics properties by Sutton. Outcrops were found underground at Eagle Point and the Q parameters for RQD, J_n, J_r, and J_a were assessed. The outcrops mapped were correlated to the A/R values obtained previously from the headings by the geology staff. For example, using the equation for Q' as found in section 3.2.1, a Q' of 1.2 was calculated for the A5/R1 geological classification. A similar procedure was carried through for each classification. Properties for the other three categories (A5/R1, A3/R2, and A1/R3) were inferred by Sutton and Milne (1998) because these combinations are noted by the geology staff, but exposures were not available at the time of the study. Parameters for the Q' classification system, including RQD, J_n, J_r, and J_a, were assessed and linked to the geology assessment of the rock mass, as shown in Table 5-2.

Table 5-2 Results of rock mass classification, Q', input parameters compared to geological classification categories (After Sutton, 1998). Number of areas mapped for the classification values are also included.

Geology Classification	Number of areas mapped	Q' Parameters				Q'
		RQD %	Joint Set Number, J_n	Joint Roughness, J_r	Joint Alteration, J_a	
A5 / R1	Data Inferred	50	4	0.75	8	1.2
A7 / R1	2	20	4	0.75	10	0.4
A3 / R2	Data Inferred	90	9	1.5	4	3.8
A5 / R2	1	75	7.5	1.5	6	2.5
A7 / R2	3	60	6	1.5	6	2.5
A1 / R3	Data Inferred	100	9	2	1	22
A3 / R3	4	100	9	1.5	1.5	11
A5 / R3	1	100	9	1.5	2	8.3
A7 / R3	1	90	9	1.5	4	3.8

In an internal document amendment (Appendix B), Sutton and Milne (1998) differentiated between gneissic and pegmatoidal rock types for the R3 rock strength categories. The gneissic rock types are defined as the rock types which are highly foliated, and this category includes the biotite-quartz-feldspar gneiss, graphitic gneiss, and quartz-feldspar gneiss rock types. The pegmatoidal rock types are generally more massive, with larger block sizes than the gneissic rock types. The pegmatoidal rock type category encompasses the other rock types, which include pegmatite, feldspar porphyry, quartzite, and calc-silicate gneiss. The calc-silicate gneiss is included in the pegmatoidal rock type category as there are fewer exposures of this rock type, and the engineering properties of this rock type are similar to the other pegmatoidal rock types. The pegmatoidal rock type term is used for all of these rock types as it is the most common rock type found that is not gneissic. The updates to the individual Q' input parameters for these are shown in Table 5-3.

Sutton's correlation between the Q' rock mechanics classification system and the geology assessment of R1 to R3 and A1 to A7 rock types is summarized in Table 5-4. The largest variation in Q' was found to be related to the geology rock strength categories, R1 to R3. The A1 through A7 alteration codes primarily act to influence the joint alteration rating in the stronger R2 and R3 categories (Forster et al., 2006). The mine currently uses this classification system based on the alteration and strength of the rock. The Q' values have been used to estimate hanging wall dilution for many stopes.

Table 5-3 Results of rock mass classification, Q', input parameters compared to rock type and geological R3 classification categories (after Sutton and Milne, 1998)

Rock Type	Geology Classification	Q' Parameters				Q'
		RQD %	Joint Set Number, J_n	Joint Roughness, J_r	Joint Alteration, J_a	
Pegmatoidal	A1 / R3	100	9	2	1	22.2
	A3 / R3	100	9	2.3	1	25.6
	A5 / R3	100	9	2.3	2	12.8
	A7 / R3	90	9	2.3	4	5.8
Gneissic	A1 / R3	100	9	1.5	1	16.7
	A3 / R3	100	9	1.5	1.5	11.1
	A5 / R3	100	9	1.5	2	8.3
	A7 / R3	90	9	1.5	4	3.8

Table 5-4 Correlation between the R1 to R3 and A1 to A7 geology system and Q' classification systems (After Sutton and Milne, 1998). Q' values for each alteration and rock strength category are shown.

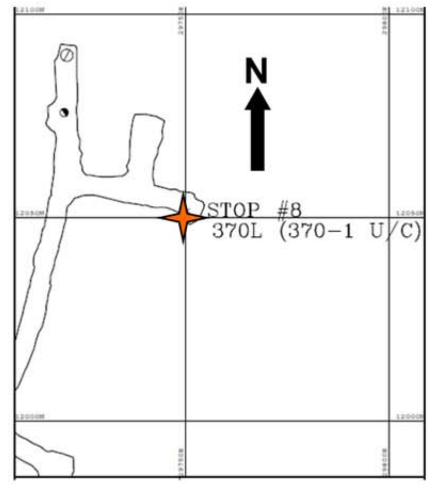
Alteration	Rock Strength			
	R1 (very weak)	R2 (weak)	R3 (medium strong)	
			Gneissic	Pegmatoidal
A1 (fresh)	N/A	N/A	16.7	22.2
A3 (weak)	N/A	3.8	11.1	25.6
A5 (moderate)	1.2	2.5	8.3	12.8
A7 (strong)	0.4	2.5	3.8	5.8

5.4.2 **Mapping Data from External Audits**

The site employs an external rock mechanics consultant, Dr. Rimas Pakalnis, to audit the rock mechanics procedures at the mine site. Dr. Pakalnis routinely completes rock mass classification assessments during his audits, and an example from a report (Cameco Corporation Internal Document, 2012) is shown in Figure 5-7. In this figure, a photo of the area visited, a description of the area and observations, an RMR_{76} rating and the rating input values, and a map showing the specific area mapped are all shown. For this particular example, dashed white lines were included on the photograph to indicate two joint sets which have the potential to form wedges in the back of the drift.

The observations were tabulated by each RMR_{76} parameter, and a summary of the observations may be found in Table 5-5. This data was collected from 10 site visit reports conducted between 2003 and 2012 (Cameco Corporation Internal Document, 2003, 2005, 2006, 2007a, 2007b, 2008a, 2008b, 2009, 2010, and 2012). When a range of values was indicated for a particular parameter, an average of the range was assumed for the rating.

The map location for each RMR_{76} observation shown in Table 5-5 was used to determine the rock type and A/R value gathered previously during routine mine geological drift mapping for each rating. An example of the site geological mapping which has been digitized into the mine design program is shown in Figure 5-8. The information for the area shown corresponds to the excerpt from the audit report in Figure 5-7. The red star corresponds to the location where the RMR_{76} rating was completed. The figure includes the surveyed dimensions of the drift, the rock type, geological structures and their orientations, the site geology A/R values, and the dates for the completion of the development advance. All RMR_{76} observations and corresponding geological maps may be found in Appendix C.



PROXIMITY OF THE ORE FW. HAVE A SPAN OF 5m WITH RMR OF 55%. CONCERN IS "DEAD WEIGHT" STRUCTURE, HOWEVER, SHOWN WITH SUPPORT IN PLACE WHICH IS 2.4m LONG #6 REBAR ON 1.2m X 1.2m PATTERN THE AREA IS STABLE. HAVE QUARTZ VUGS IN BACK (SOURCE OF WATER).

RMR BACK/WALL GNEISS

1) STRENGTH	100-200MPa	12
2) RQD	75%-50%	13
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT-SLT	20-12
5) GRNWTR	DRY	10
	RATING(WALL)	72%-62%
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	55%

STOP #8: 370L, ORE ACCESS TO 370-1 STOPE U/C. 163 ZONE MINED FROM 370L-340L

Figure 5-7 Excerpt from Dr. Pakalnis' report (Cameco Corporation Internal Document, 2012) showing the typical information and observations included in the consultant reports. Dashed white lines on the photograph indicate potentially wedge-forming joint sets.

Table 5-5 Summary of RMR_{76} observations and parameter ratings (After Cameco Corporation Internal Document, 2003, 2005, 2006, 2007a, 2007b, 2008a, 2008b, 2009, 2010, and 2012).

Case No.	Level	Strength		RQD		Spacing		Condition		Groundwater		RMR_{76}
		Observation	Rating	Observation	Rating	Observation	Rating	Observation	Rating	Observation	Rating	
1	270L (290-505 O/C)	100-200+ MPa	12	90%+	17	0.3m-1.0m	15	Tight	20	Dry	10	**74**
2	420L (420-060 U/C)	10-25 MPa	2	50%+	8	50mm-0.3m	12	Graphite and clay	6	Dry	10	**38**
3	420L (X/C #1)	50-100 MPa	7	50-75%	13	50mm-0.3m+	15	Tight	16	Dry	10	**61**
4	420L (420-045 U/C)	3-10 MPa	1	<25%	5	50mm+	8	Gouge	8	Dry	10	**32**
5	420L Main Haulage	100-200 MPa	12	90-100%	17	0.3-1m+	20	Tight	20	Dry	10	**79**
6	400L (420-020 O/C)	50-100 MPa	7	50-75%	10	50mm-0.3m	10	Tight	13	Dry	10	**50**
7	400L Main Haulage	50-100 MPa	7	50-75%	15	50mm-0.3m	15	Tight	15	Dry	10	**62**
8	320L (330-090 O/C)	50-100 MPa	7	50-75%	13	50mm-0.3m+	13	Tight/weak	13	Dry	10	**56**
9	180L (210-550 O/C)	100-200 MPa	12	75-90%	17	50mm-0.3m+	15	Tight	20	Dry	10	**74**
10	90L? (120-350B O/C)	25-50 MPa	4	50-75%	13	0.3-1m	15	Tight/Gouge	9	Damp	7	**48**
11	90L (120-350B O/C)	25-50 MPa	4	50-75%	13	0.3-1m	15	Tight/Gouge	9	Damp	7	**48**
12	90L (105-350AX O/C)	50-100 MPa	7	75-90%	17	50mm-0.3m+	15	Tight/SLT	12	Dry	10	**61**
13	245L (260-950 O/C)	50-100 MPa	7	50-75%	12	50mm-0.3m+	15	Tight/SLT	12	Dry	10	**56**
14	385L (400-075 O/C)	25-50 MPa	4	50-75%	13	50mm-0.3m+	15	Tight/SLT	9	Dry	10	**51**
15	385L (385-050 O/C)	25-50 MPa	4	<25%	3	<50mm	5	Gouge	6	Dry	10	**28**
16	385L Ramp Decline	25-50 MPa	6	25-50%	8	50mm-0.3m	10	Tight/SLT	9	Dry	10	**43**
17	355L (370-055 Access)	25-50 MPa	4	50-75%	13	0.3m+	17	Graphitic	9	Dry	10	**53**
18	215L (230-950 O/C)	100-200 MPa	12	90%	17	0.3m+	17	Tight/SLT	16	Dry	10	**72**
19	202L (202-1 U/C)	100-200 MPa	12	75-90%	17	0.3m+	15	Tight	20	Dry	10	**74**
20	202L (Waste Develop.)	100-200 MPa	12	75-90%	17	50mm-0.3m	12	Tight/SLT	15	Dry	10	**66**
21	90L (105-513 O/C)	100-200 MPa	12	75-50%	13	50mm-0.3m+	15	Tight/SLT+	16	Dry	10	**66**
22	102L (Access drift)	100-200 MPa	12	75-90%	15	0.3m	15	Tight	20	Wet	7	**69**
23	170L (170-095 U/C)	100-200 MPa	12	75-50%	13	50mm-0.3m+	15	Tight	20	Dry	10	**70**
24	170L (185-085 O/C)	100-200 MPa	12	75-50%	10	50mm-0.3m	10	SLT+	12	Dry	10	**54**
25	125X DD Bay	100-200 MPa	12	75-50%+	13	50mm-0.3m+	15	Tight-SLT+	16	Dry	10	**66**
26	90L (105-533 O/C)	100-200 MPa	12	75-50%+	13	50mm-0.3m+	15	Tight-SLT+	16	Dry	10	**66**
27	275L (163Z Access)	100-200 MPa	12	90-75%	17	50mm-0.3m+	15	Tight-SLT	13	Mod Press	4	**61**
28	122L EAR Access	100-200 MPa	12	90-75%	17	50mm-0.3m+	15	Tight	20	Damp	7	**71**
29	105L (Bulkhead #4)	100-200 MPa	12	90-75%	17	50mm-0.3m+	15	Tight	20	Dry-Severe	5	**69**
30	105L (Bulkhead #3)	100-200 MPa	12	90-75%	17	50mm-0.3m+	15	SLT	12	Dry-Severe	2	**58**
31	115L (FW 125-075 O/C)	100-200 MPa	12	75-50%	13	50mm-0.3m+	13	SLT	12	Dry	10	**60**
32	115L (HW 125-075 O/C)	100-200 MPa	12	90-75%	17	50mm-0.3m+	15	SLT	12	Dry	10	**66**
33	90L (Bulkhead #1)	100-200 MPa	12	90-75%	17	50mm-0.3m+	15	Tight-SLT	16	Mod-Severe	2	**62**
34	90L (Bulkhead #2)	100-200 MPa	12	90-75%	17	50mm-0.3m+	15	Tight	20	Dry	10	**74**
35	272L (292-1A/B O/C)	100-200 MPa	12	90-75%	17	50mm-0.3m+	12	Tight-SLT	16	Dry	10	**67**
36	80L (HW 100-085 O/C)	100-200 MPa	12	75-50%	13	50mm-0.3m+	15	Tight-SLT	16	Dry	10	**66**
37	80L (BK 100-085 O/C)	100-200 MPa	11	75-50%	13	50mm-0.3m+	15	Tight-SLT	16	Dry	10	**65**
38	80L (FW 100-085 O/C)	100-200 MPa	12	75-50%	13	50mm-0.3m+	15	Tight-SLT	16	Dry	10	**66**
39	125L (EXHAUST DRIFT)	100-200 MPa	12	75-50%	13	50mm-0.3m+	15	Tight-SLT	16	Dry	10	**66**
40	272L (272-1 U/C)	100-200 MPa	12	75-50%	13	50mm-0.3m+	15	Tight-SLT	12	Dry	10	**62**
41	82L Ramp (Back)	100-200 MPa	12	75-50%	13	50mm-0.3m+	12	Tight-SLT	16	Dry-Damp	8	**61**
42	82L Ramp (Clay gouge)	25-50 MPa	5	50-25%	8	50-300mm	10	SLT	12	Dry-Damp	8	**43**
43	102L FW Drift	100-200 MPa	12	75-50%	13	50mm-0.3m+	15	Tight	20	Dry-Moist	8	**68**
44	275L (163Z Access)	25-50 MPa	4	50-25%	5	50mm+	8	Gouge	6	Dry	10	**33**
45	270L 02NFW Access	100-200 MPa	12	75-50%	13	300mm+	10	Tight-SLT	16	Dry	10	**61**
46	170L (144Z Access)	100-200 MPa	12	90-75%	17	0.3-1m-	17	Tight	20	Wet	7	**73**
47	155L (170-085 O/C)	100-200 MPa	12	75-50%	13	50mm-0.3m	10	Tight-SLT	16	Dry	10	**61**
48	252L (272-2 O/C)	100-200 MPa	12	75-50%	13	50mm-0.3m+	15	SLT	12	Damp/Drip	7	**59**
49	280L (280-075 U/C)	100-200 MPa	12	90-75%	17	50mm-0.3m+	15	SLT	12	Dry	10	**66**
50	362L Access Ramp	100-200 MPa	12	90-75%	17	50mm-0.3m+	15	Tight	20	Dry	10	**74**
51	80L (144Z Access)	100-200 MPa	12	75-50%	13	50mm-0.3m	10	Tight-SLT	16	Damp	7	**58**
52	100L (125-695 O/C)	25-50 MPa	4	50-25%	8	50mm-0.3m+	12	SLT-Open (clay)	9	Dry-Damp	8	**41**
53	180L (144Z Intersect.)	50-100 MPa	7	50-25%	8	50mm-0.3m-	7	SLT-Open	9	Dry	10	**41**
54	180L (180-870 U/C)	100-200 MPa	12	90-75%	17	50mm-0.3m+	15	Tight	20	Dry	10	**74**
55	180L (180-870 U/C)	25-50 MPa	4	50-25%	8	<50mm	5	SLT, Open	9	Damp	7	**33**
56	150L (144Z Ramp)	100-200 MPa	12	90-75%	17	50mm-0.3m+	15	Tight	20	Damp	7	**71**
57	252L (272-5/6 O/C)	100-200 MPa	12	75-50%+	15	50mm-0.3m+	15	Tight-SLT	16	Damp	7	**65**
58	342L Ramp	50-100 MPa	7	90-75%-	15	50mm-0.3m+	15	SLT	12	Damp	7	**56**
59	272L FW Exhaust	100-200 MPa	12	90-75%-	17	50mm-0.3m+	15	Tight-SLT	16	Dry	10	**70**
60	82L Access Drift	100-200 MPa	12	90-75%-	17	50mm-0.3m+	15	Tight-SLT	16	Dry	10	**70**
61	230L 163Z Exh Drift	100-200 MPa	12	90-75%-	17	50mm-0.3m+	15	SLT	12	Dry	10	**66**
62	150L (170-880/900 O/C)	25-50 MPa	4	90-75%	17	50mm-0.3m+	15	SLT-Open (clay)	9	Dry	10	**55**
63	150L (170-870 O/C)	25-50 MPa	4	90-75%	17	50mm-0.3m+	15	SLT-Open (clay)	9	Dry	10	**55**
64	80L (144Z Remuck)	100-10 MPa	6	75-50%-	13	50mm-0.3m	10	Tight-Open	13	Damp-Wet	5	**47**
65	100L (125-675 O/C)	50-10 MPa-	5	75-50%-	13	50mm-0.3m	10	SLT	12	Damp	7	**47**
66	90L (02 Zone)	100-200 MPa	12	90-75%	17	50mm-0.3m+	15	SLT	12	Dry	10	**66**
67	80L - 144 Zone	25-50 MPa	3	50-25%	6	50-300mm-	8	OPN-GOUGE	3	Moist	7	**27**
68	80L - 144 Zone	50-100 MPa	7	75-50%	13	50-300mm+	15	SLT	12	Dry	7	**54**
69	125L (140-860 O/C)	50-100 MPa	7	75-50%	13	50-300mm+	15	Tight-SLT	16	Dry	10	**61**
70	151L Access Drift	100-200 MPa	12	75-50%	13	50-300mm+	13	Tight	20	Dry	10	**68**
71	370L (370-1 U/C)	100-200 MPa	12	75-50%	13	50-300mm+	15	TIGHT-SLT	16	Dry	10	**66**
72	272L (292-2 O/C)	100-200 MPa	12	75-50%	13	50-300mm+	13	Tight	16	Dry	7	**61**

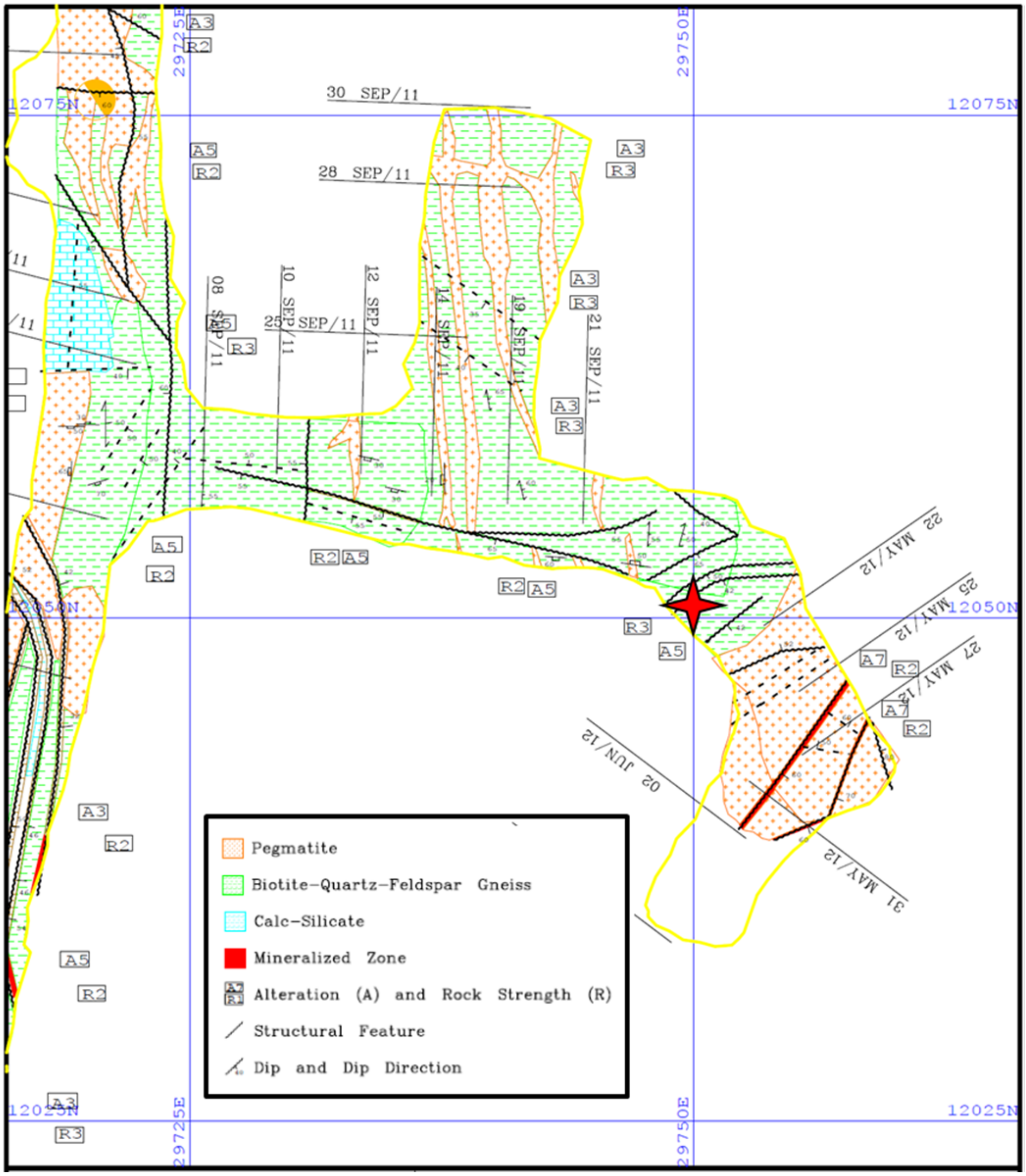


Figure 5-8 Geological mapping of the same area as the example shown in Figure 5-7.

A total of 72 RMR_{76} observations (Cameco Corporation Internal Document, 2003, 2005, 2006, 2007a, 2007b, 2008a, 2008b, 2009, 2010, and 2012) were compared to the geological mapping results collected by the geology staff for each of those areas, as shown in Table 5-6. In circumstances where there was a variety of lithologies indicated, the rock type description provided in the report and/or the predominant rock type from the geology mapping was used. The joint orientation adjustments to the RMR_{76} values, previously presented in Table 2-5 and Table 2-6, were included with each RMR_{76} observation to account for the influence of discontinuity orientation on the stability of the drift opening. In all cases for this study, the RMR_{76} observations stated are the unadjusted observations. In the report excerpt presented in Figure 5-7, the unadjusted RMR_{76} observation is 72-62, the adjustment is -10 for an unfavourable discontinuity orientation, and the weighted adjusted RMR_{76} value is 55 for the drift back. The weighted average is influenced by the observer's experience and impressions of the rock mass.

The average RQD values for the two rock types and each of the A/R classifications are presented in Table 5-7. Also shown are the average RMR_{76} observations for each of the categories and the number of observation ratings which the values are based upon. The A3/R2 category for the pegmatoidal rock types had no observations, so the values for it were interpolated.

Table 5-6 Comparison between RMR_{76} ratings to rock type and site geological classification.

No.	Location	RMR_{76}	Rock Type	A/R Rating
1	270L (290-505 O/C)	74	Gneiss	A5/R2
2	420L (420-060 U/C)	38	Pegmatite	A5/R2
3	420L (X/C #1)	61	Pegmatite	A5/R3
4	420L (420-045 U/C)	32	Pegmatite	A5/R1
5	420L Main Haulage	79	Pegmatite	A5/R3
6	400L (420-020 O/C)	50	Pegmatite	A5/R2
7	400L Main Haulage	62	Pegmatite	A3/R3
8	320L (330-090 O/C)	56	Pegmatite	A5/R2
9	180L (210-550 O/C)	74	Gneiss	A3/R3
10	90L? (120-350B O/C)	48	Gneiss	A7/R1
11	90L (120-350B O/C)	48	Gneiss	A7/R1
12	90L (105-350AX O/C)	61	Pegmatite	A5/R2
13	245L (260-950 O/C)	56	Gneiss	A5/R2
14	385L (400-075 O/C)	51	Pegmatite	A5/R2
15	385L (385-050 O/C)	28	Pegmatite	A5/R2
16	385L Ramp Decline	43	Pegmatite	A5/R2
17	355L (370-055 Access)	53	Pegmatite	A5/R2
18	215L (230-950 O/C)	72	Gneiss	A5/R2
19	202L (202-1 U/C)	74	Gneiss	A5/R3
20	202L (Waste Develop.)	66	Gneiss	A5/R2
21	90L (105-513 O/C)	66	Gneiss	A5/R2
22	102L (Access drift)	69	Gneiss	A3/R3
23	170L (170-095 U/C)	70	Gneiss	A3/R3
24	170L (185-085 O/C)	54	Gneiss	A3/R3
25	125X DD Bay	66	Pegmatite	A3/R3
26	90L (105-533 O/C)	66	Gneiss	A5/R2
27	275L (163Z Access)	61	Quartzite	A5/R2
28	122L EAR Access	71	Pegmatite	A1/R3
29	105L (Bulkhead #4)	69	Gneiss	A3/R3
30	105L (Bulkhead #3)	58	Gneiss	A7/R2
31	115L (FW 125-075 O/C)	60	Gneiss	A5/R2
32	115L (HW 125-075 O/C)	66	Gneiss	A7/R1
33	90L (Bulkhead #1)	62	Gneiss	A3/R2
34	90L (Bulkhead #2)	74	Gneiss	A3/R3
35	272L (292-1A/B O/C)	67	Gneiss	A3/R2
36	80L (HW 100-085 O/C)	66	Gneiss	A5/R2
37	80L (BK 100-085 O/C)	65	Gneiss	A5/R2
38	80L (FW 100-085 O/C)	66	Gneiss	A5/R2
39	125L (EXHAUST DRIFT)	66	Pegmatite	A1/R3
40	272L (272-1 U/C)	62	Gneiss	A3/R2
41	82L Ramp (Back)	61	Gneiss	A5/R2
42	82L Ramp (Clay gouge)	43	Pegmatite	A3/R3

Table 5-6 (Cont.) Comparison between RMR_{76} ratings to rock type and site geological classification.

No.	Location	RMR_{76}	Rock Type	A/R Rating
43	102L FW Drift	68	Gneiss	A3/R3
44	275L (163Z Access)	33	Gneiss	A5/R2
45	270L 02NFW Access	61	Gneiss	A1/R3
46	170L (144Z Access)	73	Pegmatite	A3/R3
47	155L (170-085 O/C)	61	Gneiss	A5/R3
48	252L (272-2 O/C)	59	Gneiss	A7/R2
49	280L (280-075 U/C)	66	Gneiss	A5/R2
50	362L Access Ramp	74	Gneiss	A3/R3
51	80L (144Z Access)	58	Pegmatite	A3/R3
52	100L (125-695 O/C)	41	Pegmatite	A7/R1
53	180L (144Z Intersect.)	41	Gneiss	A5/R1
54	180L (180-870 U/C)	74	Gneiss	A5/R?
55	180L (180-870 U/C)	33	Gneiss	A5/R?
56	150L (144Z Ramp)	71	Pegmatite	A5/R2
57	252L (272-5/6 O/C)	65	Gneiss	A5/R2
58	342L Ramp	56	Gneiss	A7/R1
59	272L FW Exhaust	70	Gneiss	A5/R3
60	82L Access Drift	70	Gneiss	A3/R3
61	230L 163Z Exh Drift	66	Gneiss	A3/R2
62	150L (170-880/900 O/C)	55	Pegmatite	A7/R2
63	150L (170-870 O/C)	55	Pegmatite	A5/R2
64	80L (144Z Remuck)	47	Pegmatite	A5/R2
65	100L (125-675 O/C)	47	Pegmatite	A5/R2
66	90L (02 Zone)	66	Gneiss	A3/R3
67	80L - 144 Zone	27	Pegmatite	A7/R1
68	80L - 144 Zone	54	Gneiss	A7/R1
69	125L (140-860 O/C)	61	Gneiss	A7/R1
70	151L Access Drift	68	Pegmatite	A5/R2
71	370L (370-1 U/C)	66	Gneiss	A5/R3
72	272L (292-2 O/C)	61	Pegmatite	A7/R2

Table 5-7 Average RQD and RMR_{76} values for each of the A/R classifications for pegmatoidal and gneissic rock types.

Rock Type	A/R Classification	Average $RQD_{RMR\ 76}$	Average RMR_{76}	No. of Observations
Pegmatoidal	A7/R1	38	34.0	2
	A5/R1	25	32.0	1
	A7/R2	73	58.0	2
	A5/R2	63	52.1	14
	A3/R2	71**	60**	0**
	A5/R3	79	70.0	2
	A3/R3	62	60.4	5
	A1/R3	73	68.5	2
Gneissic	A7/R1	69	55.5	6
	A5/R1	38	41.0	1
	A7/R2	73	58.5	2
	A5/R2	68	63.0	14
	A3/R2	78	64.3	4
	A5/R3	73	67.8	4
	A3/R3	77	68.8	10
	A1/R3	63	61.0	1

***No exposures were mapped. Values inferred.*

5.4.3 **Mapping Data Completed Internally**

RMR_{89} observations have been collected by a variety of site personnel, as shown in Table 5-8. The results for these observations were also compared to the geology A/R rating and rock type, and the site mapping data sheets may be found in Appendix D. The observational ratings were generally for the better quality rock types. There were many other rock mass classification values collected, but this data lacked the specific location within the mine, which made it impossible to compare to the rock type and A/R value.

Table 5-8 Site personnel RMR_{89} observations (Obs.).

Case No.	Level	Strength		RQD		Spacing		Condition		Groundwater		RMR_{89}	Rock Type	A/R
		Obs.	Rating	Obs.	Rating	Obs.	Rating	Obs.	Rating	Obs.	Rating			
1	382L Ramp	100-250 MPa	12	95%	20	0.1 - 1.0 m	12	Chlorite <1mm thick	20	Damp	15	79	Granitic Dome	A1/R3
2	370L - 163 Zone	100-200 MPa	12	90%	20	0.3 to 0.5 m	10	Tr clay & Tr Chlorite	20	Dry	15	77	Pegmatoid	A1/R3
3	272L EAR #5	100-200 MPa	12	90%	17	0.1 - 0.5 m	10	Clean to tr	19	Dry	15	73	Gneiss	A3/R3
4	322L (02 Next Plug #4)	50 - 100 MPa	10	90%	17	0.1 to 0.5 m	10	Clean to tr	21	Dry	15	73	Gneiss and Porphyry	A5/R2 to A3/R3
5	272X Concrete Plug	100 MPa	10	90%	17	0.2 - 0.5 m	10	Mod alt, clay/gouge seam	10	Wet	7	54	Porphyry	A3/R2
6	232L Concrete Plug (up ramp)	100 - 200 MPa	12	90%	17	0.1 - 0.5 m	10	Clean to tr	18	Dry	15	72	Gneiss	A3/R3
7	232L Concrete Plug (down ramp)	100-200 MPa	12	90%	17	0.1 - 0.5 m	10	Clean to tr	18	Dry	15	72	Gneiss	A3/R3
8	272X Remuck	150 MPa	12	90%	17	0.1 - 0.5 m	10	Clay	16	Dry	15	70	Pegmatoid	A3/R3
9	302L TTA	100-200 MPa	12	90%	17	0.3 - 1.0 m	12	Clean	25	Dry	15	81	Pegmatoid	A1/R3
10	302L Ramp & Remuck	100-200 MPa	12	90%	17	0.3 - 1.0 m	12	Trace chlorite	20	Dry	15	76	Pegmatoid	A3/R3
11	272L EAR #4 acc and remuck	100-200 MPa	12	90%	17	0.3 - 1.0 m	12	Trace chlorite	20	Dry	15	76	Granitic Dome	A1/R3

5.4.4 **Data Analysis**

Figure 5-9 is a graphical summary of all of the sources of rock mass classification which have a corresponding geological A/R rating. The A/R classification values are arranged along the horizontal axis according to their relative quality, from the weakest strength and alteration combination to the strongest. The vertical axis is the RMR_{76} value. The RMR_{76} observations completed by Dr. Pakalnis were graphically compared to the corresponding geological mapping A/R classifications and rock type. The Sutton Q' measurements and inferred values were converted to an equivalent RMR_{76} value using the $RMR_{76} = 9\ln Q + 44$ formula and plotted by the corresponding A/R classification and rock type. The site engineering personnel RMR_{89} observations were converted to RMR_{76} values by using the descriptions recorded for the observations and the RMR_{76} classification system. Hoek et al. (1995) state that the minimum RMR_{76} observational value is 18, and the corresponding minimum RMR_{89} value is 23. Assuming these minimum values to be equivalent for the two Rock Mass Rating systems, the corresponding RMR_{89} value is noted beside the RMR_{76} value on the vertical scale.

For lower quality rock masses, the results in Figure 5-9 indicate that a pegmatoidal rock type has a much lower RMR_{76} rating than a gneissic rock type of the same A/R rating. It is possible to compare the Sutton Q' parameter assumptions to the Rock Mass Rating (1976 and 1989) data points by A/R classification and rock type by looking at the individual parameters that comprise the classification estimates.

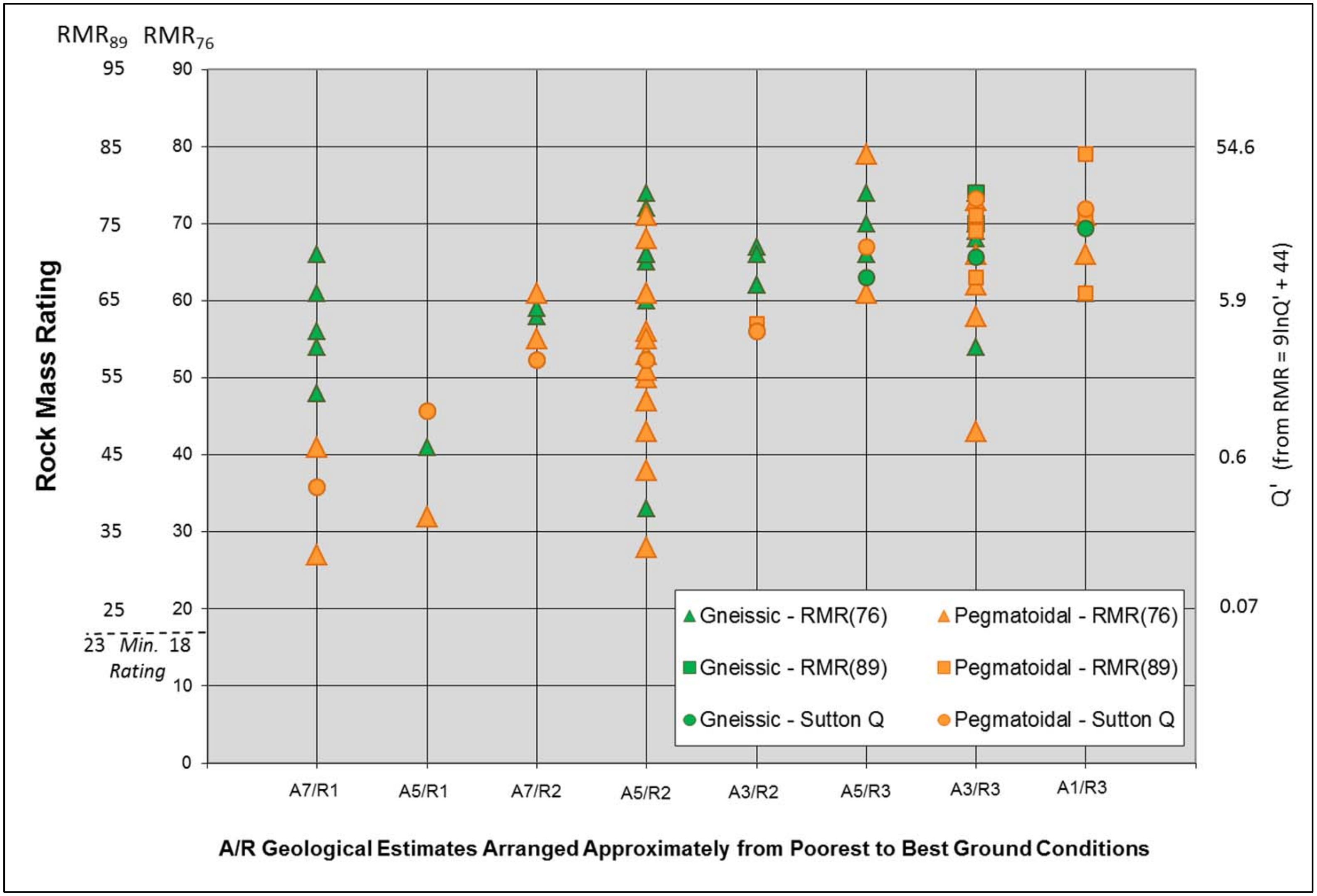


Figure 5-9 Graphical comparison of RMR_{76} observations, site personnel RMR_{89} observations, and Sutton Q' correlation observations to the site geology staff A/R values and rock type.

5.5 Summary

There are several sources of geological or geotechnical information available at the Eagle Point Mine; geotechnical drift mapping, geological drift mapping, and diamond drill hole core logging. There are limitations and advantages to all three methods of data collection.

Point load testing was completed on five of the main rock types found at the Eagle Point Mine. As the geological A/R classification system is subjective, it was hoped that the point load testing results could be used to find a more objective correlation between rock strength and degree of alteration. The results indicated a general decrease in the overall rock strength with an increase in the degree of alteration.

Previous work was completed by Sutton and Milne (1998) to link Q' input parameters to geological A/R classifications. Site personnel collect rock mass classification data, however, this data was not collected with the intention of relating the rock mass classification values with the geological A/R classifications, and the specific locations of the mapping were not recorded in most cases. The site employs an external rock mechanics consultant who collects Rock Mass Rating (RMR_{76}) observations, and these observations have locations specified and were compared to the geology strength and alteration (A/R) ratings.

The rock types were separated into two main groups; gneissic rock types, and pegmatoidal rock types. The gneissic rock types are defined as the rock types which are highly foliated, and this category includes the biotite-quartz-feldspar gneiss, graphitic gneiss, and quartz-feldspar gneiss rock types. The pegmatoidal rock type category encompasses the other rock types, which include pegmatite, feldspar porphyry, quartzite, and calc-silicate gneiss.

For lower quality rock masses, a pegmatoidal rock type has a much lower RMR_{76} rating than a gneissic rock type of the same A/R rating. Forming a link between the individual rock mass properties that define rock mass classification should improve the overall correlation and is presented in the next chapter.

Chapter 6 - Improved Approach for Correlating Rock Classification Data and Systems

6.1 Individual Parameter Correlations

Although all of the rock mass classification systems presented have elements in common, it is not possible to directly link each of the systems. Approximations, such as Equation 2.4, have been developed, but are also considered to be coarse relations at best (Milne et al., 1998). A new approach has been developed for this project to link the individual parameters of each system.

The Rock Mass Rating systems (1976 and 1989) and Q system have some parameters in common. The RMR_{76}, RMR_{89} systems and the Q system use RQD values for their system. For both of the Rock Mass Rating systems, the RQD rating is incorporated based on a range of RQD ratings (Table 2-2 and Table 2-3), while the Q system uses a single value based on an average RQD rating for the rock mass (Table 2-7). The RMR systems consider joint condition, which includes descriptions for joint roughness and alteration. The Q system separates the joint roughness (J_r) and joint alteration (J_a) observations. All three systems include considerations for water inflow in the form of the joint water condition for the RMR systems, and the J_w parameter for the Q system.

The systems also have several parameters which are more difficult to compare. The RMR systems have a parameter for the average discontinuity spacing. The Q system joint set number, J_n, is more indicative of the average block shape and not the block size. The RMR systems include a consideration for the intact rock strength, while the Q system has a stress reduction factor, SRF, which is a reflection of the stress in the area.

Milne et al. (2013) presented a comparison of the ranges for the Q system joint alteration (J_a) and joint roughness (J_r) to the RMR_{76} joint descriptions (Figure 6-1). Error bars are included for the range of J_a and J_r values based on the RMR_{76} description values. For example, for an

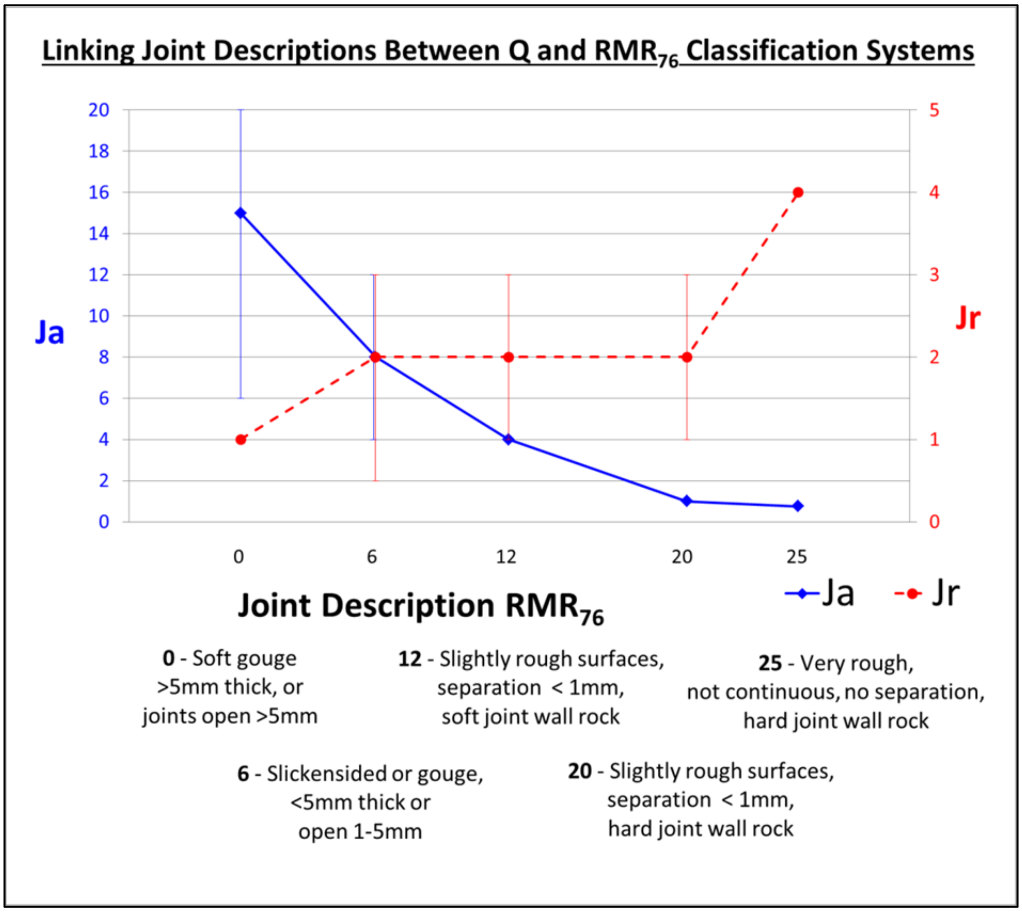


Figure 6-1 Link between RMR_{76} joint description and the Q system J_a and J_r parameters (Milne et al., 2013).

RMR_{76} joint condition description of "Slickensided or gouge, <5mm thick or open 1-5mm", the corresponding joint condition value would be 6. Based on that description, the J_a value could range from a minimum possible value of 4 (rock wall contact, softening or low-friction clay mineral coatings, discontinuous, 1-2 mm or less) to a maximum value of 12 (rock wall contact before 10 cm shear, swelling clay fillings, continuous, <5 mm thick). The J_r value could range from a minimum possible value of 0.5 (slickensided, planar) to a maximum value of 3 (rough and irregular, undulating).

A similar comparison by Milne et al. (2013) was made for the RMR_{76} joint spacing values, the RQD value, and the number of joints per cubic metre (joints/m^3) as shown in Figure 6-2.

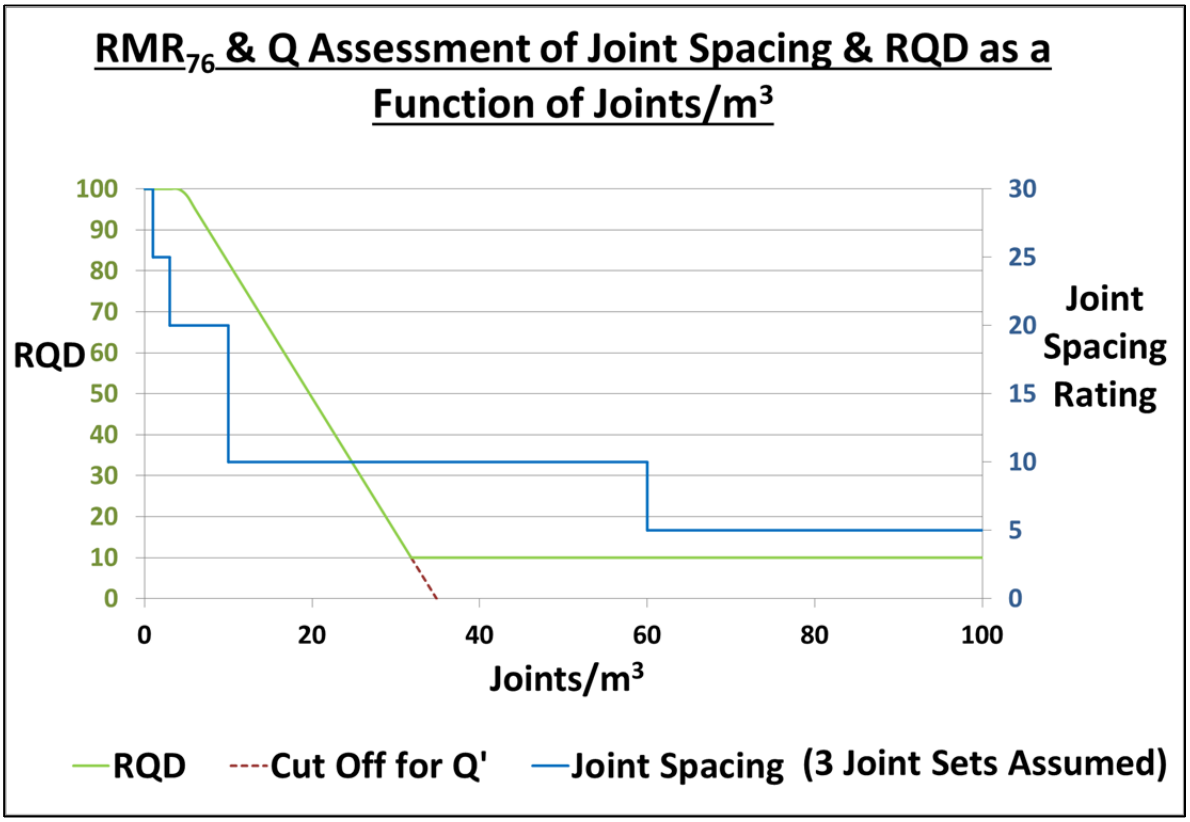


Figure 6-2 Comparison of RMR_{76} joint spacing, RQD and number of joints per cubic metre (Milne et al., 2013).

6.1.1 Comparison of RMR_{76} joint condition and Q' joint alteration (J_a) parameters

The Q' values are required in the empirical graphs for stope stability and dilution, but the rock mass classifications have been completed in the Rock Mass Rating (1976 and 1989) systems. Although there are similarities between the two classification systems, some assumptions are required to compare the values in each system. As mentioned in Section 2.2.3, the relationship between the RMR_{76} system and the Q system may be approximated using Equation 2.4. This equation is intended to be used as a coarse comparison between the two systems. The individual input parameters should still be compared separately.

The RMR_{76} joint condition parameter is approximately equivalent to a combination of the Q' J_a and J_r parameters, but the parameters from the two systems are not proportionally equivalent.

From Table 5-2 depicting the individual parameters for the Sutton Q' assumptions, there is a large range between the minimum and maximum J_a values. The joint roughness values were not considered for the comparison, as they show little variation over the total A/R range, and the RMR_{76} system includes only a very coarse assessment of roughness. The descriptions corresponding to the J_a values have been compared to the joint condition descriptions and their values in the RMR_{76} system. Equivalent J_a parameter values and RMR_{76} joint condition parameter values have been either determined from similar descriptions, or interpolated between two RMR_{76} joint condition descriptions. The equivalent RMR_{76} values for the Sutton Q' J_a parameters are found in Table 6-1.

In Figure 6-3 and Figure 6-4, the geology A/R values are plotted on the horizontal axis from the weakest to the strongest. The A/R categories are gradational, and the boundaries between the categories are subjective. The RMR_{76} joint condition parameter data points were plotted for each A/R value and rock type. The Sutton Q' J_a values were plotted adjacent to the corresponding RMR_{76} joint condition description value. The site personnel mapping joint descriptions were compared to the RMR_{76} joint condition descriptions, and these values were plotted for each equivalent RMR_{76} joint condition and geology A/R observation. The results are shown in Figure 6-3 and Figure 6-4 for pegmatoidal rock types and gneissic rock types respectively.

The Sutton Q' J_a assumptions for the pegmatoidal rock types appear similar to the joint condition descriptions tabulated for RMR_{76} classification values collected by Dr. Pakalnis. The Sutton Q' values are found within the range of each of the RMR_{76} joint condition data. However, the J_a assumptions for the gneissic rock types appear significantly lower than the joint condition data collected by Dr. Pakalnis for the highly altered, low strength rock types and high for the weakly altered, high strength rock types.

Table 6-1 Description and rating comparison between RMR_{76} joint condition and Q' J_a parameters (After Bieniawski, 1976, and Barton et al., 1974)

Q' J_a Parameter Description	Value	Rock Mass Rating (1976) Joint Condition Description	Value
Rock wall contact before 10 cm shear, swelling clay fillings (continuous < 5mm thick)	10	Gouge < 5mm thick, continuous joints	6
Rock wall contact before 10 cm shear, medium or low over-consolidation, softening clay mineral fillings (continuous < 5mm thick)	8	*Interpolated*	*8*
Rock wall contact before 10 cm shear, strongly over-consolidated, non-softening clay mineral fillings (continuous < 5mm thick)	6	*Interpolated*	*10*
Rock wall contact, softening or low-friction clay mineral coatings (discontinuous coatings, 1 - 2 mm or less)	4	Slightly rough surfaces, separation < 1mm, soft joint wall rock	12
Rock wall contact, slightly altered joint walls, non-softening mineral coatings, sandy particles, clay-free disintegrated rock, etc.	2	*Interpolated*	*16*
Transition area between J_a = 2 and J_a = 1, rock wall contact, unaltered joint walls to slightly altered joint walls	1.5	*Interpolated*	*18*
Rock wall contact, unaltered joint walls, surface staining only	1	Slightly rough surfaces, separation < 1mm, hard joint wall rock	20

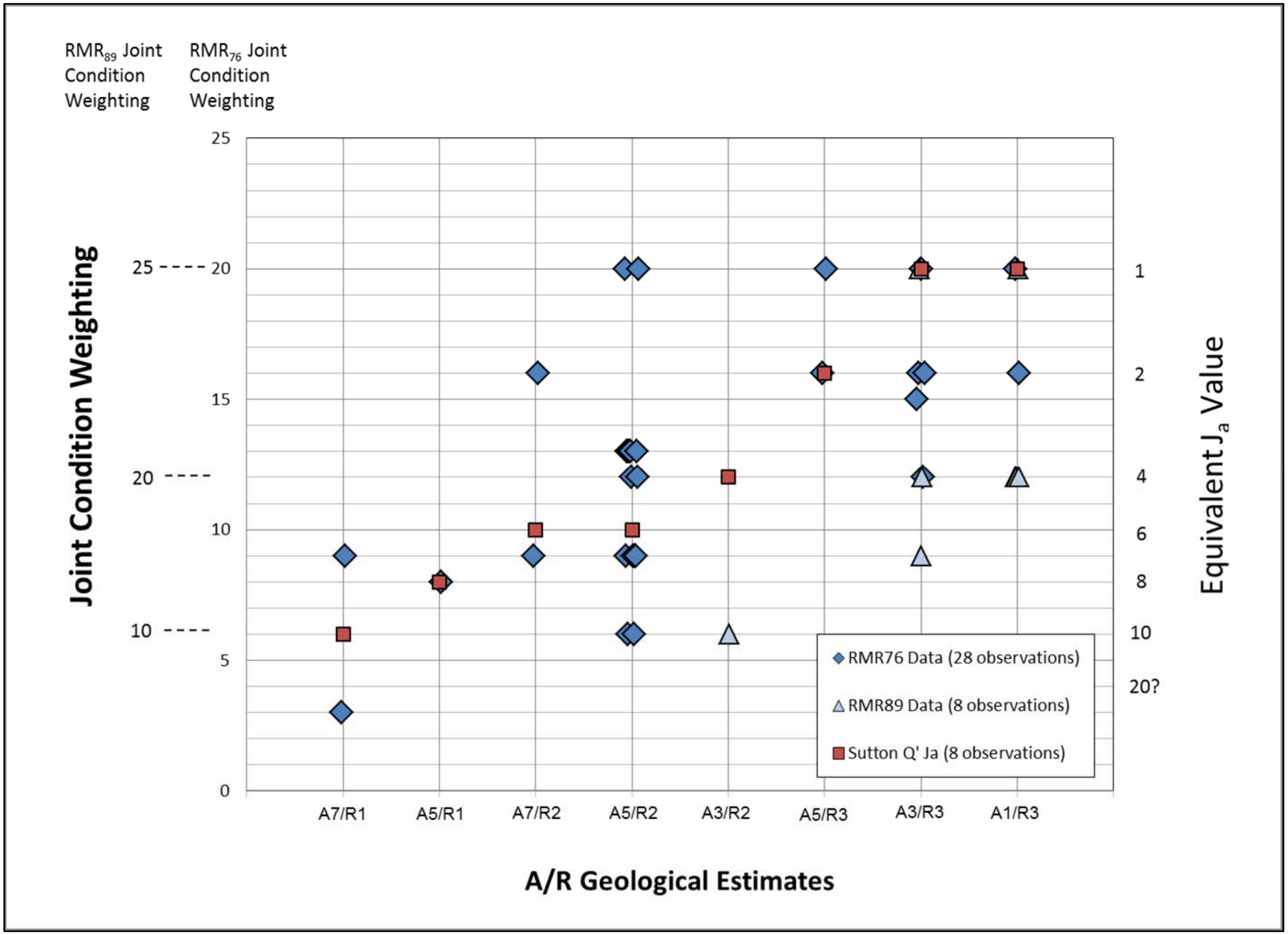


Figure 6-3 Comparison of RMR_{76} and RMR_{89} joint condition parameters to Sutton Q' J_a value for pegmatoidal rock types, arranged by A/R classification.

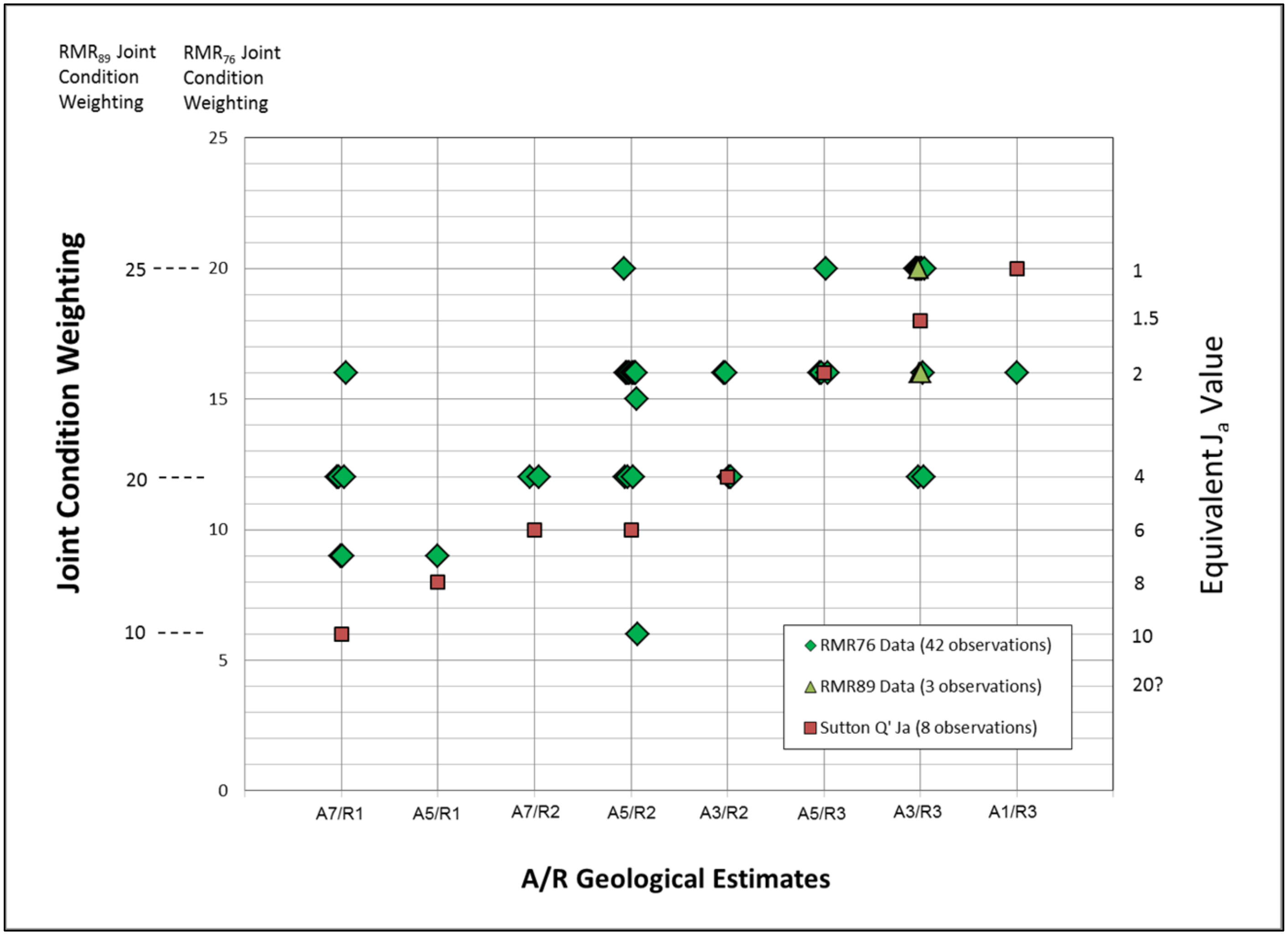


Figure 6-4 Comparison of RMR_{76} and RMR_{89} joint condition parameters to Sutton Q' J_a value for gneissic rock types, arranged by A/R classification.

If Equation 2.4 is assumed to be valid, changing the joint condition parameter value should correspond to a change in the Q' joint alteration parameter. Table 6-2 illustrates that a change in the joint condition value from 20 to 12, would result in the change in the RMR_{76} from 65 to 57. The resulting change in the Q' value using Equation 2.4, results in a Q' of 10.3 for an RMR_{76} of 65, and Q' of 4.2 for an RMR_{76} of 57. This results in a Q' reduction factor of 0.41 (from 10.3/4.2). Assuming J_a is 1 for a corresponding joint alteration of 20, applying the Q' reduction factor would result in an equivalent $J_{a(eq)}$ of 2.43. Similar calculations were completed for RMR_{76} joint condition values of 6 and 0. The resulting $J_{a(eq)}$ values are less than the parameter description correlations.

Table 6-2 Theoretical changes to Q' from RMR_{76} joint condition parameter changes.

Joint Condition	RMR_{76}	Q'	Q' Reduction Factor	J_a from RMR to Q (Equation 2.4)	*J_a From RMR Joint Description*
20	65	10.3	-	-	*1*
12	57	4.2	0.41	2.43	*4*
6	51	2.2	0.51	4.74	*8*
0	45	1.1	0.51	9.23	*15*

6.1.2 **RMR_{76} and Q' Rock Quality Designation (RQD) parameters**

A similar comparison was completed for the RMR_{76} RQD parameter weighting data points and the RMR_{89} and Sutton Q' RQD assumptions. The RMR_{76} and RMR_{89} RQD values are based upon ranges of RQD values, while the Q' system RQD values are based on specific numbers. The equivalent RMR_{76} values for the Sutton Q' RQD parameters are found in Table 6-3. The RMR_{76} and RMR_{89} RQD value weightings are identical.

The results for these comparisons are shown in Figure 6-5 and Figure 6-6 for pegmatoidal rock types and gneissic rock types respectively.

Many of the Sutton Q' RQD assumptions for the pegmatoidal rock types appear to be slightly higher than the RMR_{76} RQD data points for most A/R classifications. The Sutton Q' RQD assumptions for the gneissic rock types appear to be low for the highly altered, low strength rock types and too high for the weakly altered, high strength rock types, compared to the RMR_{76} measurements.

Table 6-3 Description and rating comparison between RMR_{76} and Q' Rock Quality Designation parameters (After Bieniawski, 1976, and Barton et al., 1974)

Q' RQD Parameter Description	Value	Rock Mass Rating (1976) RQD Parameter Description	Value
Rock Quality Designation = 20%	20	< 25% = 3	3
Rock Quality Designation = 50%	50	25% to 50% = 8; and 50% to 75% = 13	10
Rock Quality Designation = 60%	60	50% to 75% = 13	12
Rock Quality Designation = 75%	75	50% to 75% = 13; and 75% to 90% = 17	15
Rock Quality Designation = 90%	90	75% to 90% = 17; and 90% to 100% = 20	18
Rock Quality Designation = 100%	100	90% to 100% = 20	20

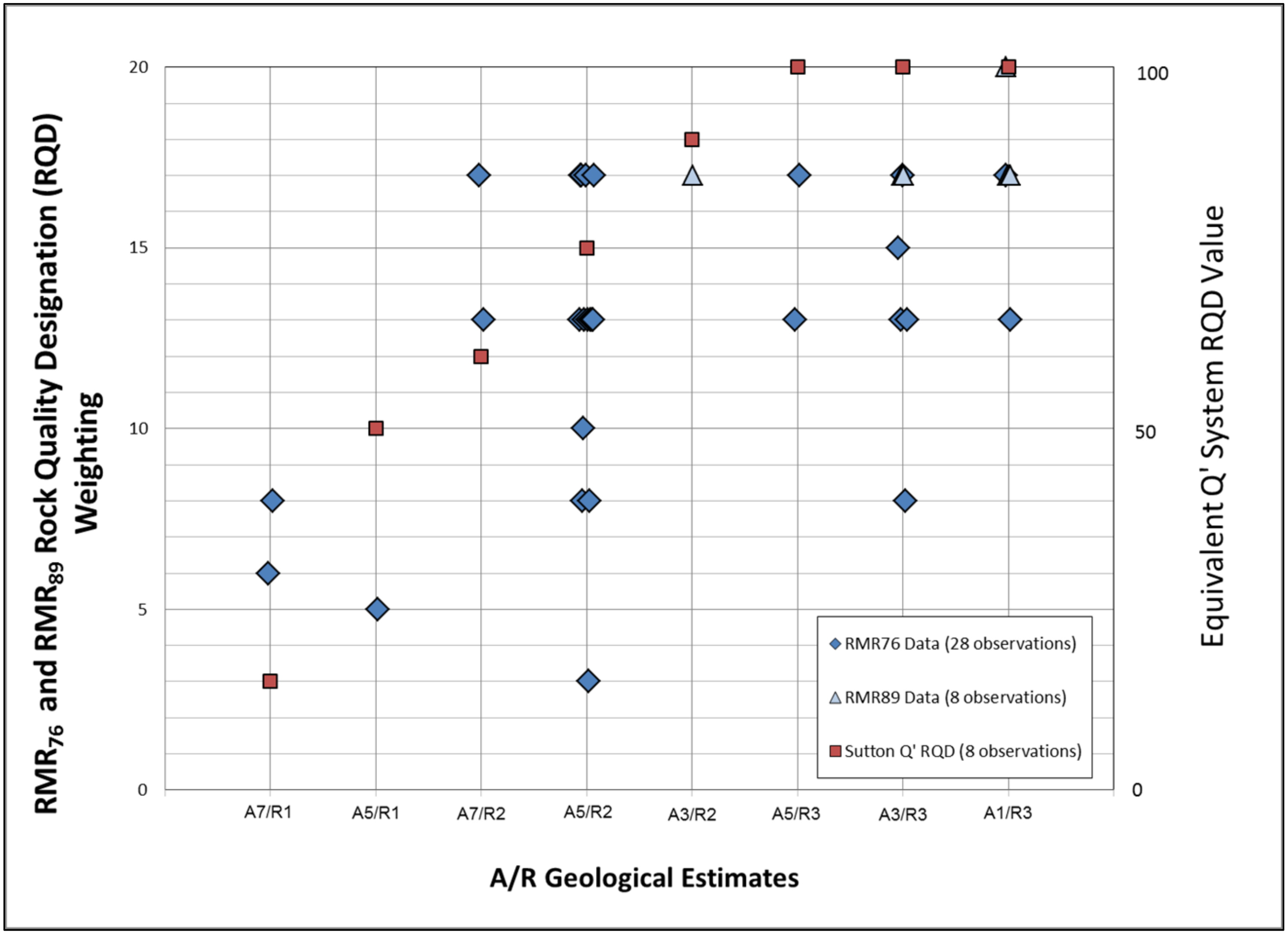


Figure 6-5 Comparison of RMR_{76} and RMR_{89} RQD parameter to Sutton Q' RQD value for pegmatoidal rock types, arranged by A/R classification.

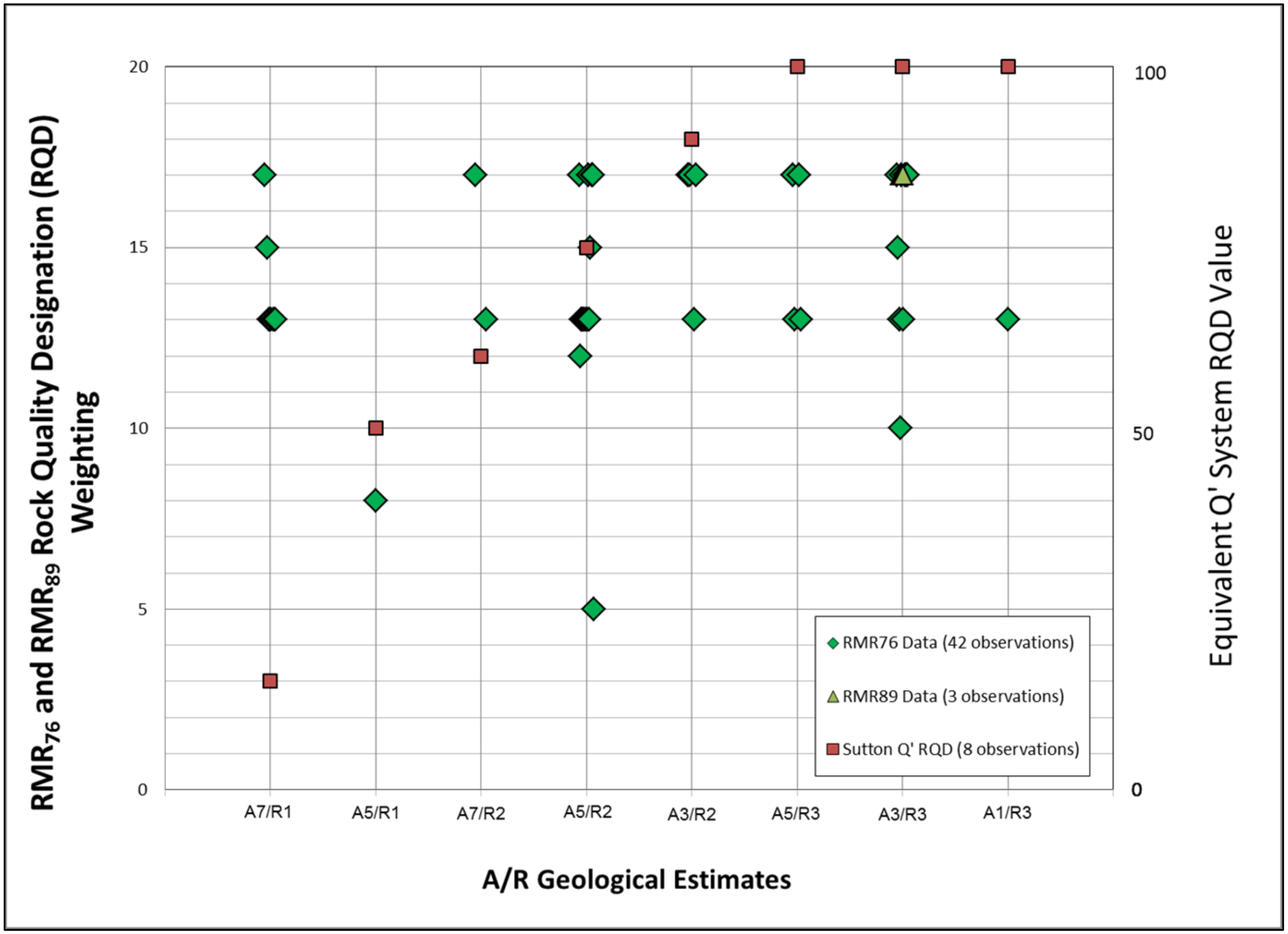


Figure 6-6 Comparison of RMR_{76} and RMR_{89} RQD parameter to Sutton Q' RQD value for gneissic rock types, arranged by A/R classification.

6.1.3 **RMR_{76} joint spacing and Q' joint set number (J_n) parameters**

Data was analyzed to see if there is any relationship between the joint spacing of the rock mass at the Eagle Point Mine and the number of joint sets of that rock mass. The RMR_{76} and RMR_{89} joint spacing parameter weighting data points were compared to the Sutton Q' J_n assumptions. There is no apparent direct correlation between the number of joint sets for a rock mass and the average joint spacing, so the A/R categories were arranged along the horizontal axis from poorest to best quality rock, and the Sutton Q' J_n values for each were noted below the A/R category. The results for these comparisons are shown in Figure 6-7 and Figure 6-8 for pegmatoidal rock types and gneissic rock types respectively.

Although there is no apparent correlation between the number of joint sets for a rock mass and the average joint spacing, presumably the more joint sets there are, the closer the average distance between joints would be. The results shown in Figure 6-7 and Figure 6-8 contradict this assumption. A lower RMR_{76} joint spacing rating indicates closer average joint spacing, and a lower Q J_n value indicates fewer joint sets for the rock mass. For the pegmatoidal rocks, a closer joint spacing was found for the poorer quality A/R categories, and the joint set number, J_n, values for these categories assume fewer joint sets for the weaker rock masses. This is opposite to what seems to intuitively be true, and may be caused by the highly sheared zones in the rock mass having more random jointing which is more difficult to separate into sets. A highly altered, clay-filled and deformed zone is challenging to categorize within the context of rock classification. Marinos et al. (2005) state, "The quantification processes used [for rock classification] are related to the frequency and orientation of discontinuities and are limited to rock masses in which these numbers can easily be measured. The quantifications do not work well in tectonically disturbed rock masses in which the structural fabric has been destroyed." An understanding of the geology is important for appropriate classification.

For the gneissic rock types as shown in Figure 6-8, there are no apparent trends in the joint spacing ratings and the A/R categories.

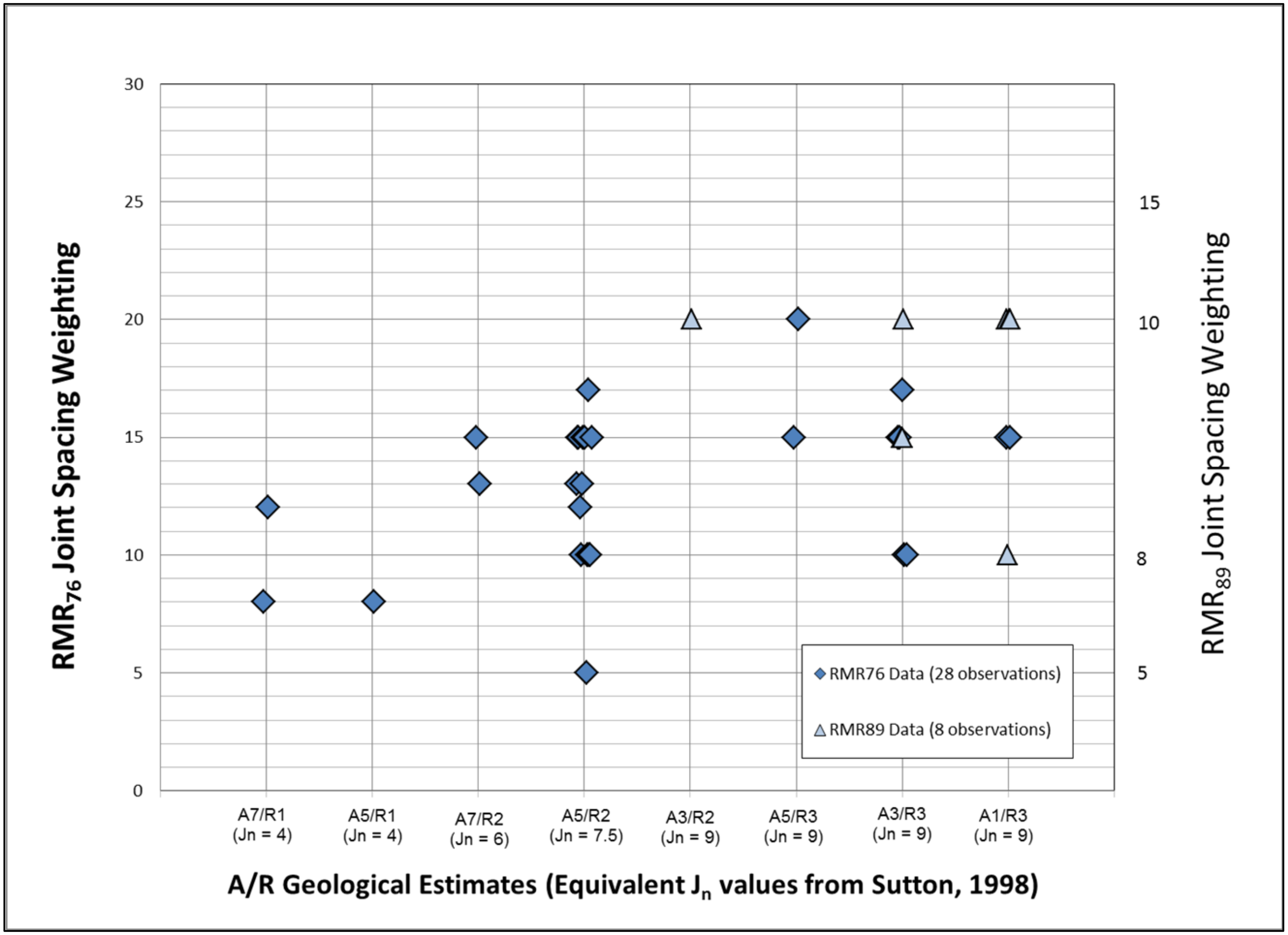


Figure 6-7 Comparison of RMR$_{76}$ and RMR$_{89}$ Joint Spacing parameter to Sutton Q' J_n parameter for pegmatoidal rock types, arranged by A/R classification.

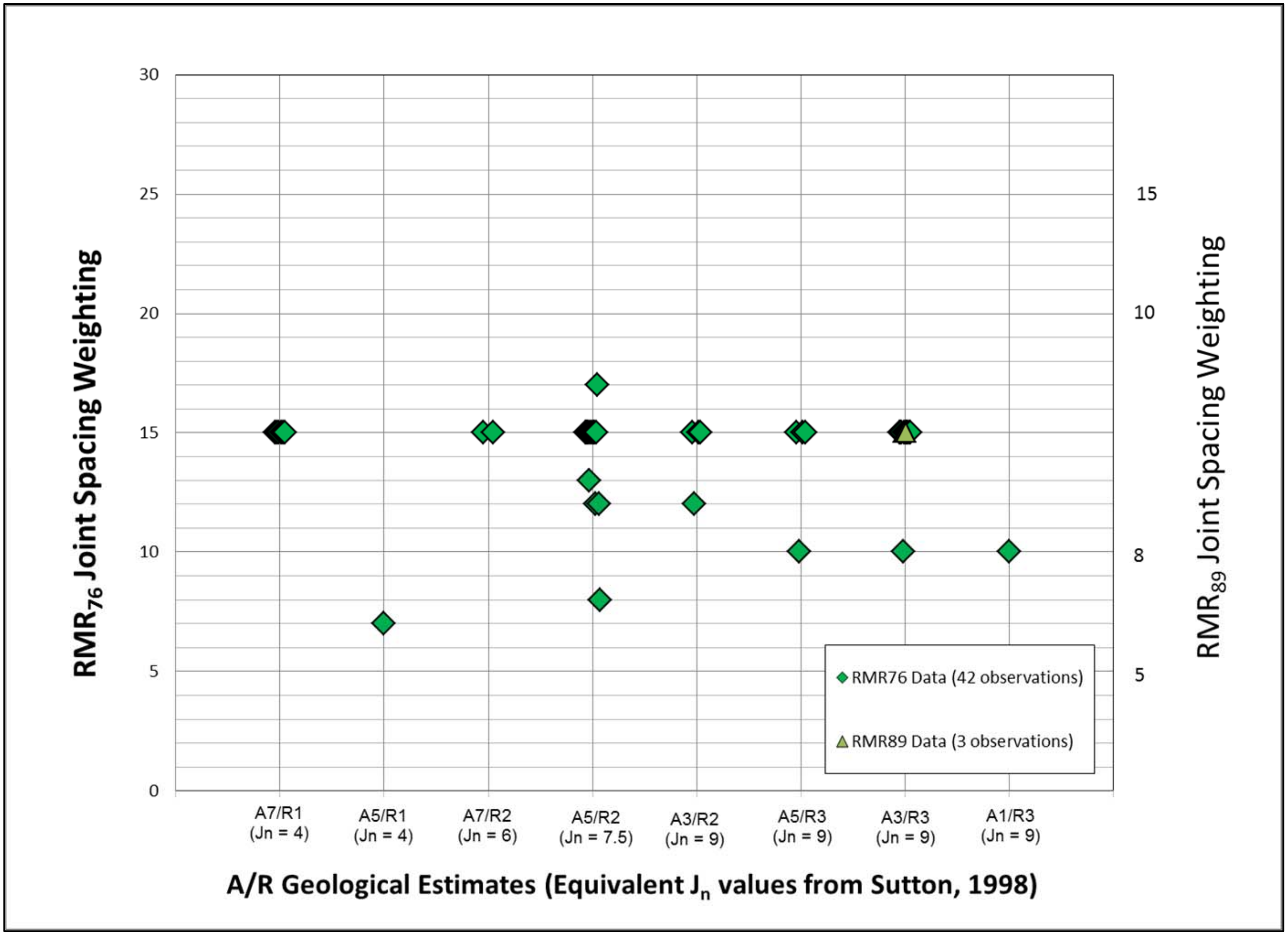


Figure 6-8 Comparison of RMR_{76} and RMR_{89} Joint Spacing parameter to Sutton Q' J_n parameter for gneissic rock types, arranged by A/R classification.

6.2 A/R System to Q' Correlation Improvements

6.2.1 **Comparison of all sources of data**

For the A/R System to Q' correlation, the most comprehensive source of information is the RMR_{76} values from the External Ground Control Audit reports (Cameco Corporation Internal Document, 2003, 2005, 2006, 2007a, 2007b, 2008a, 2008b, 2009, 2010, 2012). Other geotechnical mapping observations from several different personnel, including the author, were further incorporated into the summary values. The A7/R3 category was removed from the analysis, as this indicates a strongly altered, moderately strong rock, and there are no observations for this alteration and strength combination.

The Q' values were modified from the original Sutton (1998) work based on the RMR_{76} parameter ratings and descriptions, and the comparison charts discussed in section 5.4. The results of the updated parameters for both of the rock types are shown in Table 6-4. The changes for the pegmatoidal rock types included lowering the RQD parameter, and changing two of the J_a parameters based on the observations of the joint condition ratings in Figure 6-3. The RMR_{76} observations for the pegmatoidal A3/R3 category (Figure 6-3) show a majority of equivalent J_a values ranging from 1 to 4, indicating that a J_a value of approximately 2 could be more appropriate than the value of 1 chosen in the tables. However, the RMR_{76} observations did not take into consideration the geologist mapping approach for determining fresh (A1) to weak (A3) to moderate (A5) alteration. The data suggest a J_a assessment of 1 agrees with an expected progression of reduced alteration from A3/R2 (J_a of 3), A5/R3 (J_a of 2), A3/R3 (J_a of 1) to A1/R3 (J_a of 1). As well, any ranges specified for the RMR_{76} joint condition observations were averaged for this project to obtain the ratings. Subjectivity is expected in both the RMR_{76} and geology A/R categories, but the underlying intent is to show increased alteration from the A1 to A5 categories, which can be expected to be most pronounced on fractures (Figure 4-8).

Table 6-4 Comparison of the previous Sutton Q' parameter ratings, and the proposed Parameter Based Q' Ratings. Changes are highlighted with blue shading.

Pegmatoidal				Gneissic			
Q' Parameter	A/R Classification	Sutton Rating	Parameter Based Rating	Q' Parameter	A/R Classification	Sutton Rating	Parameter Based Rating
RQD	A5/R1	50	40	RQD	A5/R1	50	75
	A7/R1	20	25		A7/R1	20	75
	A3/R2	90	75		A3/R2	90	75
	A5/R2	75	60		A5/R2	75	75
	A7/R2	60	50		A7/R2	60	75
	A1/R3	100	75		A1/R3	100	75
	A3/R3	100	75		A3/R3	100	75
	A5/R3	100	75		A5/R3	100	75
J_n	A5/R1	4	4	J_n	A5/R1	4	9
	A7/R1	4	4		A7/R1	4	9
	A3/R2	9	9		A3/R2	9	9
	A5/R2	7.5	7.5		A5/R2	7.5	9
	A7/R2	6	6		A7/R2	6	9
	A1/R3	9	9		A1/R3	9	9
	A3/R3	9	9		A3/R3	9	9
	A5/R3	9	9		A5/R3	9	9
J_r	A5/R1	0.75	0.75	J_r	A5/R1	0.75	0.75
	A7/R1	0.75	0.75		A7/R1	0.75	0.75
	A3/R2	1.5	1.5		A3/R2	1.5	1.5
	A5/R2	1.5	1.5		A5/R2	1.5	1.5
	A7/R2	1.5	1.5		A7/R2	1.5	1.5
	A1/R3	2	2		A1/R3	1.5	1.5
	A3/R3	2.3	2.3		A3/R3	1.5	1.5
	A5/R3	2.3	2.3		A5/R3	1.5	1.5
J_a	A5/R1	8	8	J_a	A5/R1	8	4
	A7/R1	10	10		A7/R1	10	4
	A3/R2	4	3		A3/R2	4	2
	A5/R2	6	4		A5/R2	6	3
	A7/R2	6	6		A7/R2	6	4
	A1/R3	1	1		A1/R3	1	1
	A3/R3	1	1		A3/R3	1.5	1.5
	A5/R3	2	2		A5/R3	2	2

The changes for the gneissic rock types included consistent RQD and J_n parameter ratings for all A/R categories. The author's experience and observations at the mine site have found that the RQD values and number of joint sets do not change for the gneissic rock types for any of the alteration or strength observations. The J_a ratings for gneissic rock types were lowered, reflecting the lower alteration indicated by the observations of the joint condition ratings in Figure 6-4.

Both rock types retained the original Sutton Q' observations for the joint roughness, J_r, factor. The descriptions for joint roughness were specified within the rock strength (R) category described by Sutton and Milne (1998) as:

- R1 – Very Weak Rock (J_r = 0.75)
  Joint surfaces are often wavy, but smooth/polished or slickensided. A J_r value of 1.5 to 2.0 can be assigned, however, this value should be divided by 2 if the surfaces are polished or slickensided. (From Sutton and Milne, 1998)
- R2 – Weak Rock – Breaks with one blow (J_r = 1.5)
  Joints are commonly planar to wavy. A higher Jr value could be encountered in the pegmatite. (From Sutton and Milne, 1998)
- R3 – Medium Strong Rock – Many blows to break (J_r = 1.5 to 2.3)
  Joints are generally slightly rough in the foliated rocks, however, if the pegmatite the joints are more rough and between planar & wavy. (From Sutton and Milne, 1998)

The theoretical change in the number of joint sets is shown in Figure 6-9. The figure shows that the number and orientation of joint sets is primarily controlled by the rock type and stress field within a lithological unit. If faulting is present, the frequency of jointing and possibly joint set orientations will be strongly influenced by the distance to a fault zone. Within the fault zone, jointing sub-parallel to the fault may be present. Other joint orientations will often be highly disturbed, making joint set delineation difficult, if not impossible. This could account for the reduced number of joint sets observed for the pegmatoidal rock types with a high degree of alteration.

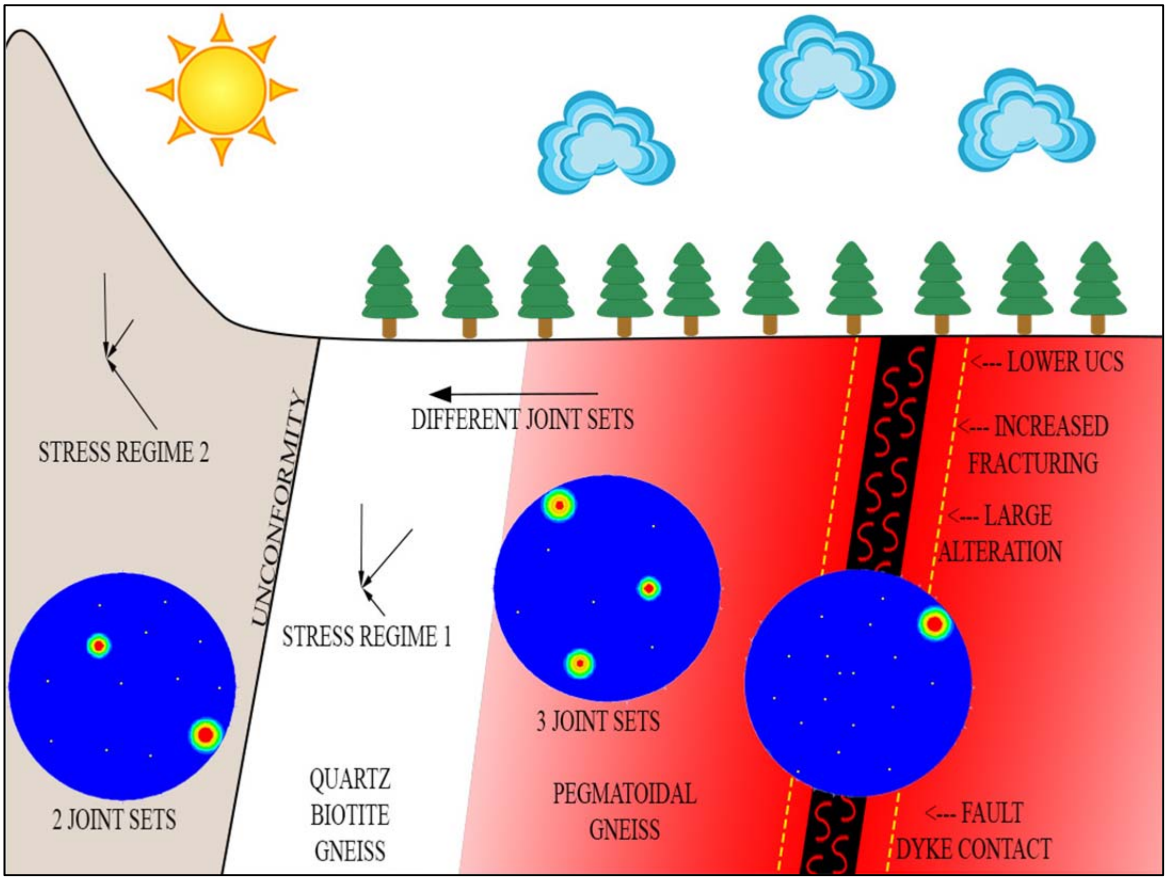


Figure 6-9 Theoretical section depicting changing numbers of joint sets and joint condition with changing stress regimes.

Another source of data which may be used for estimating the number of joint sets is geological drift mapping. Figure 6-10 depicts a stereonet of dip and dip direction measurements from geological back mapping. Two joint sets are shown, with a potential third joint set identified which may occur parallel or sub-parallel to the drift back. From this stereonet, there appears to be 3 joint sets, corresponding to a J_n value of 9.

A summary comparison of the Sutton Q' values and the Parameter Based Q' values arranged by rock type and A/R classification may be found in Table 6-5. The relative change in the Q' value is included as a percentage of the change.

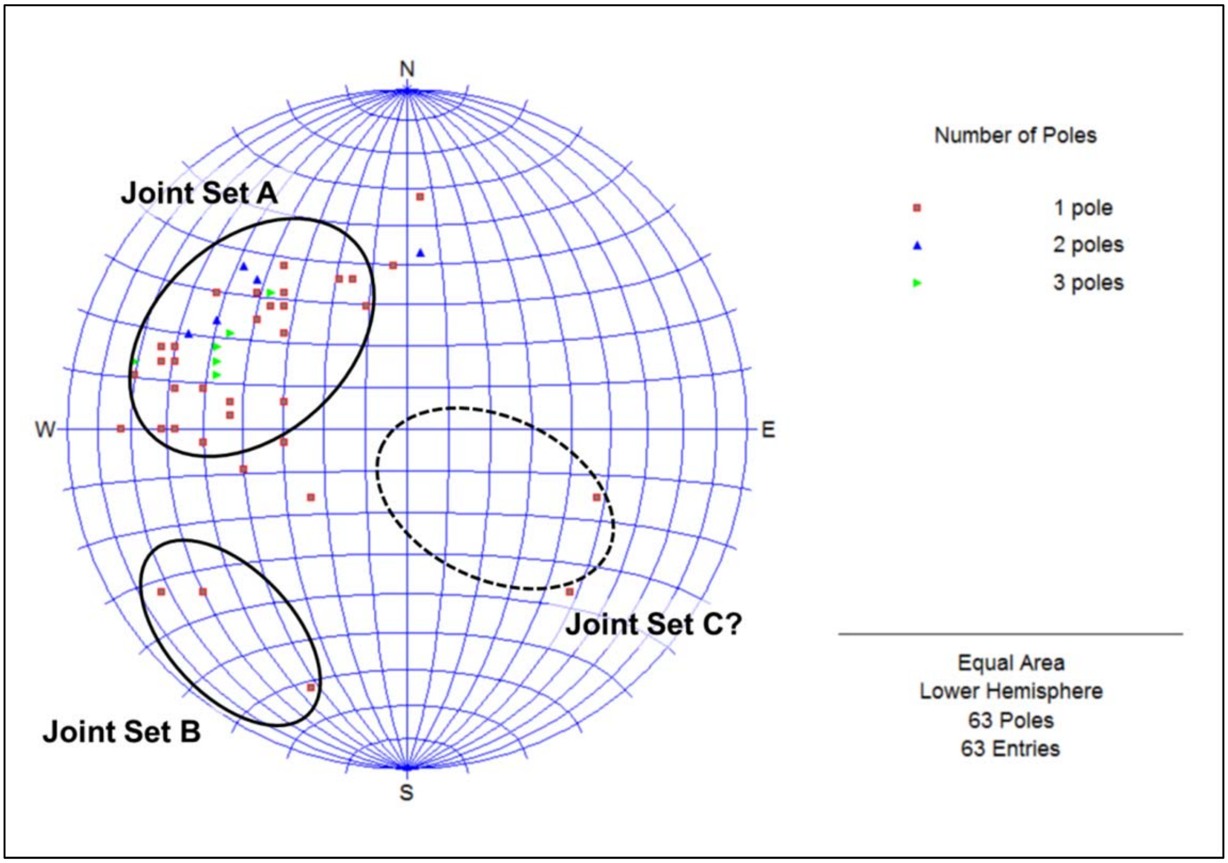


Figure 6-10 Dip and dip direction measurements from drift back mapping. Two joint sets identified with a possible third joint set parallel or sub-parallel to the drift back orientation.

Table 6-5 Comparison of Sutton Q' and Parameter Based Q' values arranged by rock type and A/R classification.

Rock Type	A/R Classification	Q'		
		Sutton	Parameter Based	% Change
Pegmatoidal	A7/R1	0.4	0.5	25%
	A5/R1	1.2	0.9	-25%
	A7/R2	2.5	2.1	-16%
	A5/R2	2.5	3.0	20%
	A3/R2	3.8	4.2	11%
	A5/R3	12.8	9.6	-25%
	A3/R3	25.6	19.2	-25%
	A1/R3	22.2	16.7	-25%
Gneissic	A7/R1	0.4	1.6	300%
	A5/R1	1.2	1.6	33%
	A7/R2	2.5	3.1	24%
	A5/R2	2.5	4.2	68%
	A3/R2	3.8	6.3	66%
	A5/R3	8.3	6.3	-24%
	A3/R3	11.1	8.3	-25%
	A1/R3	16.7	12.5	-25%

The Sutton and Parameter Based Q' values by rock type were plotted against each other in Figure 6-11 with the Sutton Q' values along the horizontal axis, and the Parameter Based Q' values along the vertical axis. A line was included to show where the Parameter Based Q' value would be equivalent to the Sutton Q' value. In general, the Parameter Based Q' values for the weaker gneissic rock masses, or those less than a Q' of 5, were increased from the Sutton values. There was little change for the pegmatoidal rock masses with a Q' less than 5.

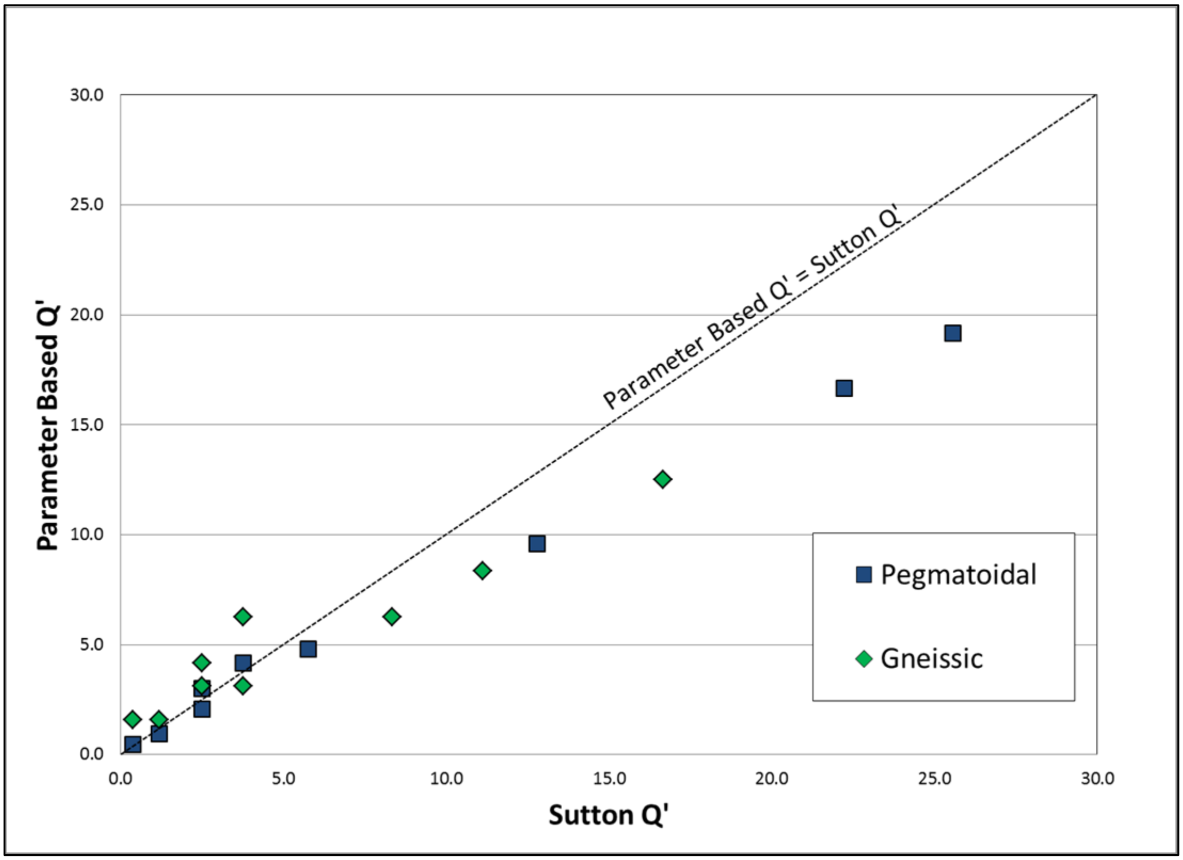


Figure 6-11 Comparison of Sutton Q' values and Parameter Based Q' values as per rock type.

The Parameter Based Q' values for the stronger rock masses, or those greater than a Q' of 5, were decreased from the Sutton values for both rock types.

The RMR_{76} values were also used to estimate a Q' value using Equation 2.4. The Sutton Q', Parameter Based Q', and RMR_{76} to Q' equation based values are presented in Table 6-6.

Table 6-6 Summary of Q' values for each A/R classification by rock type for Sutton Q', Parameter Based Q', and equivalent Q' from the RMR_{76} observations.

Rock Type	A/R Classification	Q'			
		Sutton	Parameter Based	From RMR = 9lnQ + 44	No. RMR_{76} of Ratings
Pegmatoidal	A7/R1	0.4	0.5	0.3	2
	A5/R1	1.2	0.9	0.3	1
	A7/R2	2.5	2.1	4.7	2
	A5/R2	2.5	3	2.5	14
	A3/R2	3.8	4.2	5.9	0**
	A5/R3	12.8	9.6	18.0	2
	A3/R3	25.6	19.2	6.2	5
	A1/R3	22.2	16.7	15.2	2
Gneissic	A7/R1	0.4	1.6	3.6	6
	A5/R1	1.2	1.6	0.7	1
	A7/R2	2.5	3.1	5.0	2
	A5/R2	2.5	4.2	8.3	14
	A3/R2	3.8	6.3	9.5	4
	A5/R3	8.3	6.3	14.0	4
	A3/R3	11.1	8.3	15.7	10
	A1/R3	16.7	12.5	6.6	1

***No exposures were mapped. Values inferred.*

The three Q' values for the pegmatoidal rock types were plotted by A/R classification in Figure 6-12, and for the gneissic rock types in Figure 6-13. Some of the RMR_{76} to Q' equation based values are significantly lower than the Sutton Q' and Parameter Based Q' values. This is most likely due to the fact the equation based approach is based on average conditions for correlating the two systems and cannot be expected to work in all conditions (Milne et al., 1998). As the number of RMR_{76} ratings for each A/R classification increase, the average value for the parameter based correlations should become more reliable. Additional observational data is needed for the J_r and J_n estimates.

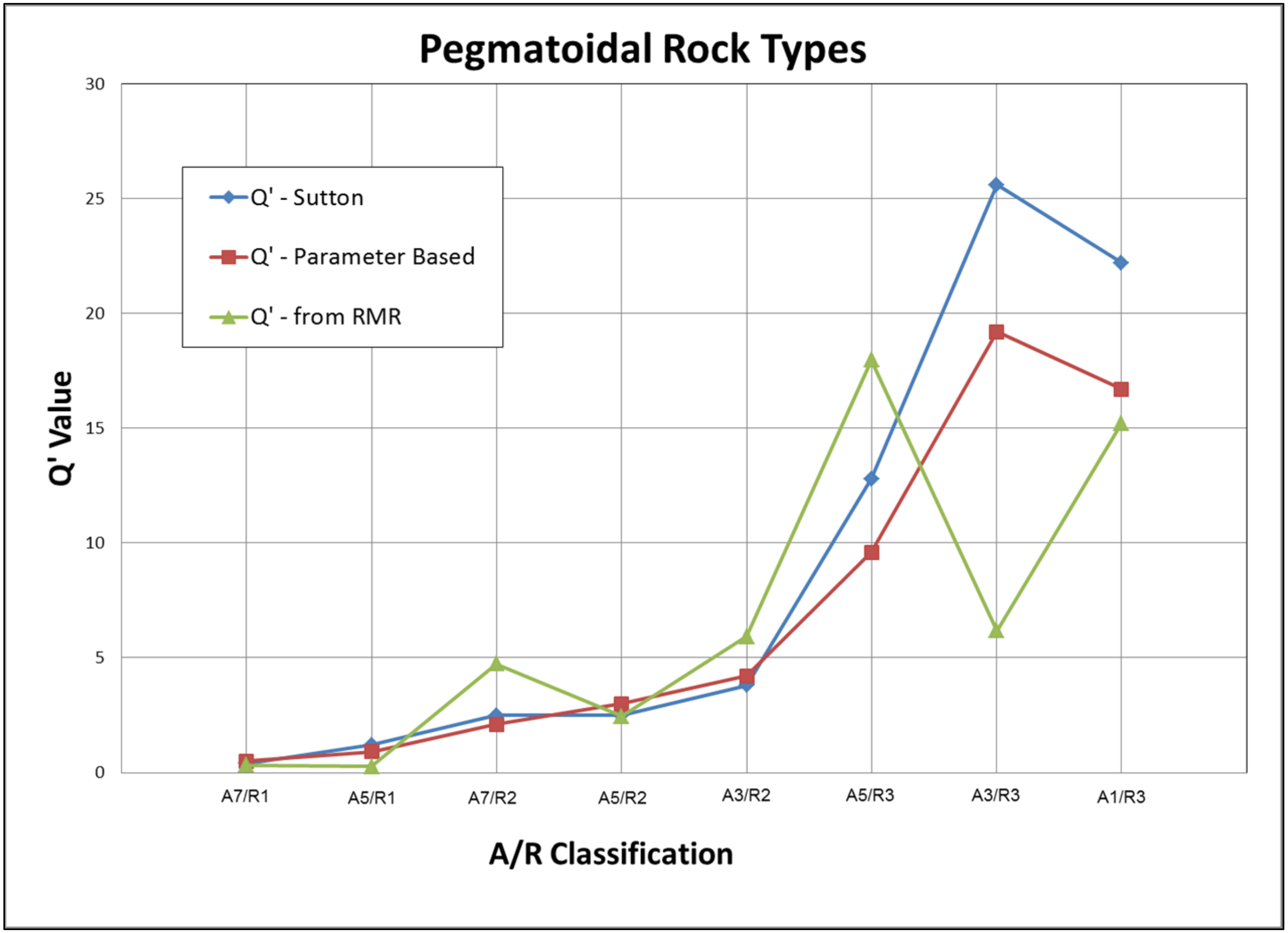


Figure 6-12 Comparison of Sutton Q', Parameter Based Q', and Q' from RMR_{76} by A/R classification for pegmatoidal rock types.

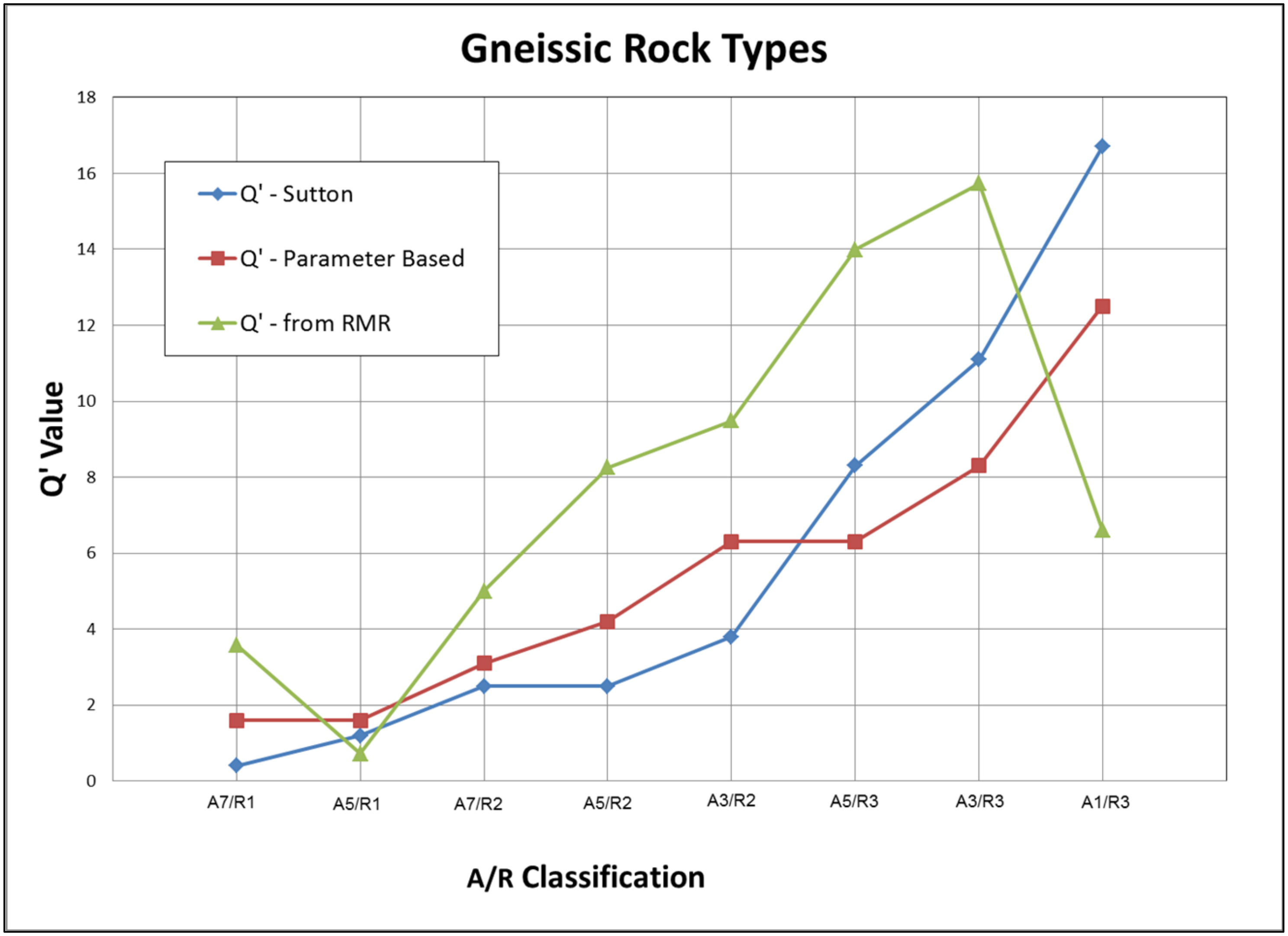


Figure 6-13 Comparison of Sutton Q', Parameter Based Q', and Q' from RMR_{76} by A/R classification for gneissic rock types.

6.3 Summary

Geological information can be a very useful tool for inferring and/or supplementing geotechnical information. Much work has been done at the Eagle Point Mine to compare geological information to rock mechanics ratings for mine design.

Multiple sources of information may be combined to provide an estimate of the rock mass classification for geotechnical design. Geological drift mapping information may contribute information on the rock type, alteration, number of joint sets, and major structures.

Because of the variability of RQD values and joint spacing within a rock mass A/R category, using Q', RMR_{76} or RMR_{89} values will not significantly improve a correlation between geotechnical classification and the A/R geology classification, as shown in Figure 6-5 and Figure 6-6. Exploration diamond drill hole core logging may provide supplementary information on RQD values. RQD is not highly dependent on the A/R category and must be assessed independently.

Geotechnical area mapping may provide additional information on the parameters which are difficult to measure due to limited access. These parameters include joint roughness, the number of joint sets, and joint infilling characteristics.

These sources of information are not a comprehensive list of the potential sources of data for rock mass classification, but they do represent the sources of information available at the Eagle Point Mine.

Further refinement on the Q' assumptions from the geological A/R values were completed based on geotechnical area mapping values from several sources, but the most valuable source of RQD information should come from diamond drill hole core logging.

Chapter 7 – Application of Geological Rock Mass Assessment for Stope Reconciliation

As shown in Sections 3.2 and 3.3, rock mass classification values can be used to attempt to predict the behaviour of an underground opening with tools such as the Modified Stability Graph and Dilution Graph. To determine the effectiveness of these tools, a methodology must be determined to collect repeatable and quantifiable measurements of rock mass properties, stress conditions, opening geometries, and subsequent rock instability if it occurs.

A methodology for combining sources of rock mass data to obtain a repeatable representative estimate of rock mass classification values has been developed. An estimate of rock mass classification values for stope design has previously been based on relatively subjective interpretations of available data. Three approaches for estimating rock classification values are applied to compare their effectiveness in predicting stope dilution.

In order to assess the performance of the prediction, there should be a repeatable method of measuring the geometry of a stope, as well as the overbreak and underbreak that may occur after mining the stope. This chapter will introduce a consistent approach for quantifying open stope geometry and dilution. A process called stope reconciliation is used to determine open stope dilution. Reconciliation between the planned and actual stope outlines is a comparison between planned and excavated stope geometries. Stope reconciliation is done after production to analyze stope performance. This is important for future production planning and for the calibration of the geotechnical assessments of rock mass performance. The analysis includes a comparison of predicted tonnage and ore grade to actual material removed and ore grade, the productivity of the mining activities, and the performance of the underground opening. The focus of this study is on the physical performance of the stope geometry, and not the productivity of the mining activities. The phrase “geotechnical stope reconciliation” is used in this study to focus on the rock mass stability assessment process.

Narrow vein mining, such as is used at the Eagle Point Mine, has its own challenges. To better understand the stope geometry, a brief overview of the stope blasting approach follows.

7.1 Stope Blasting Approach

Structurally-controlled, narrow-veined ore bodies frequently result in wandering production drifts as the mining advances along the ore. In areas of complex geology, it is particularly difficult to predict the trend of the ore vein, or in the case of multiple ore veins, which vein should be followed. Sometimes, it is only after both the overcut and undercut drifts are in place that the economic ore block may be projected. In these circumstances, this may result in an overcut drift or undercut drift which is not in the optimum location for mining. An example is provided in Figure 7-1 that shows a case where the undercut drift did not effectively follow the economic hanging wall contact for the full strike length of the stope. This creates challenges for defining the ore body geometry and for determining appropriate estimates for rock classification estimates.

The stopes are drilled and blasted in what is known as a “dice five/dice seven”, or 2:1/3:1 drilling pattern, which is common for narrow-veined orebodies. Figure 7-2 depicts an example of the dice-five/dice-seven drill hole pattern as shown in plan view. One hole from each ring is offset to provide better spatial distribution of the explosive load. In some cases an additional hanging wall or footwall hole is added to the pattern in wide ore body zones or when the undercut drift has extended too far into the footwall or hanging wall of the orebody (as shown in Figure 7-2, rings R03 to R11). The collars of the primary ring drill holes are shown as blue circles, while the secondary, or “helper”, drill holes are shown as green squares. The drill hole collars are laid out in an approximate diamond or dice pattern.

Figure 7-1 shows three section views to illustrate the dice-five/dice-seven drilling pattern. Figure 7-1a shows the primary ring drill holes, Figure 7-1b shows the drill hole in the helper ring, and Figure 7-1c shows the helper ring overlain on the primary ring. The "helper" drill hole fills the gap between the footwall drill hole and the center drill hole of the primary ring.

Due to the complex geology and difficulty in defining the economic stope limits, the blast hole furthest into the hanging wall is used to help define the hanging wall geometry for geotechnical stope reconciliation. The extent of any dilution or overbreak is measured relative to this hanging wall blast hole. Due to the location of the helper hole, the area between this hole and the cavity monitoring survey line would be greater than the area for each of the primary rings. As such, only the primary rings were analyzed for the calculation of the ELOS of each stope.

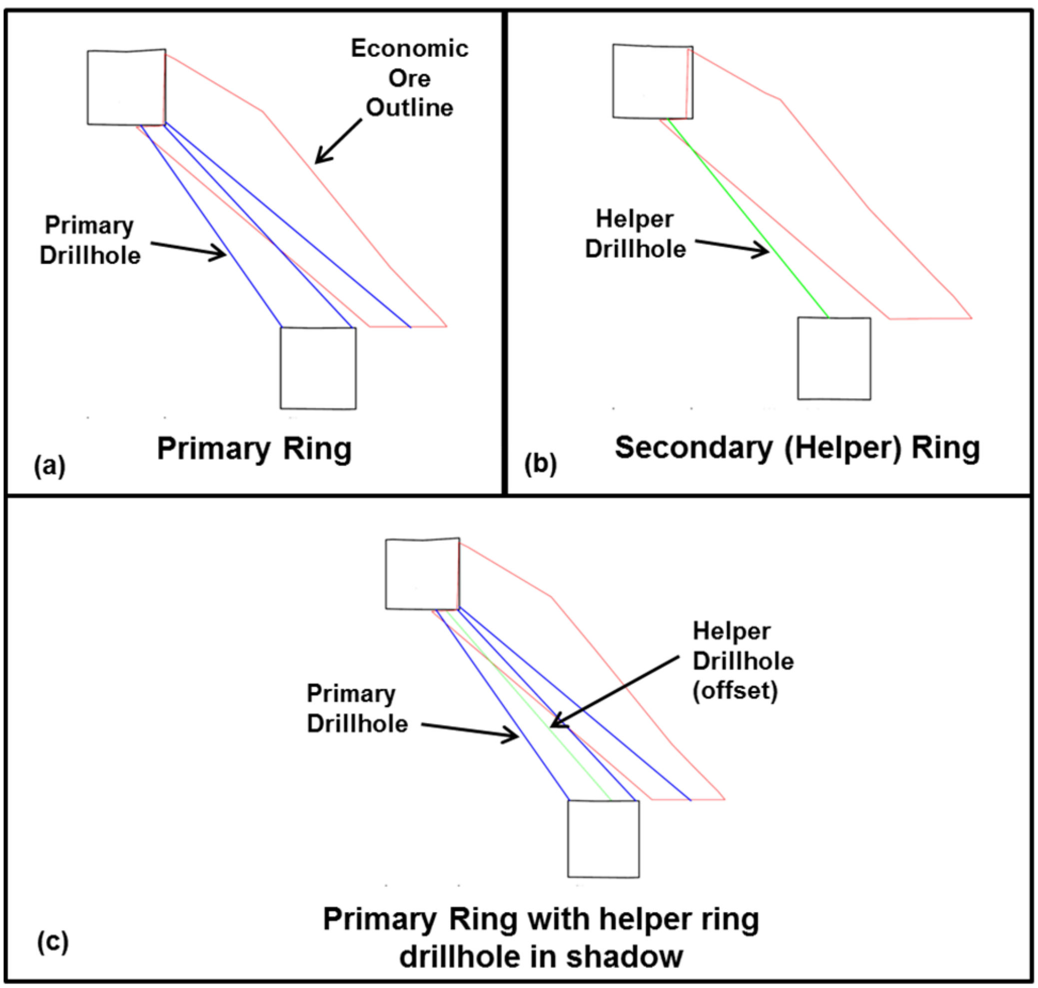


Figure 7-1 Section views illustrating the dice-five/dice-seven drilling pattern.

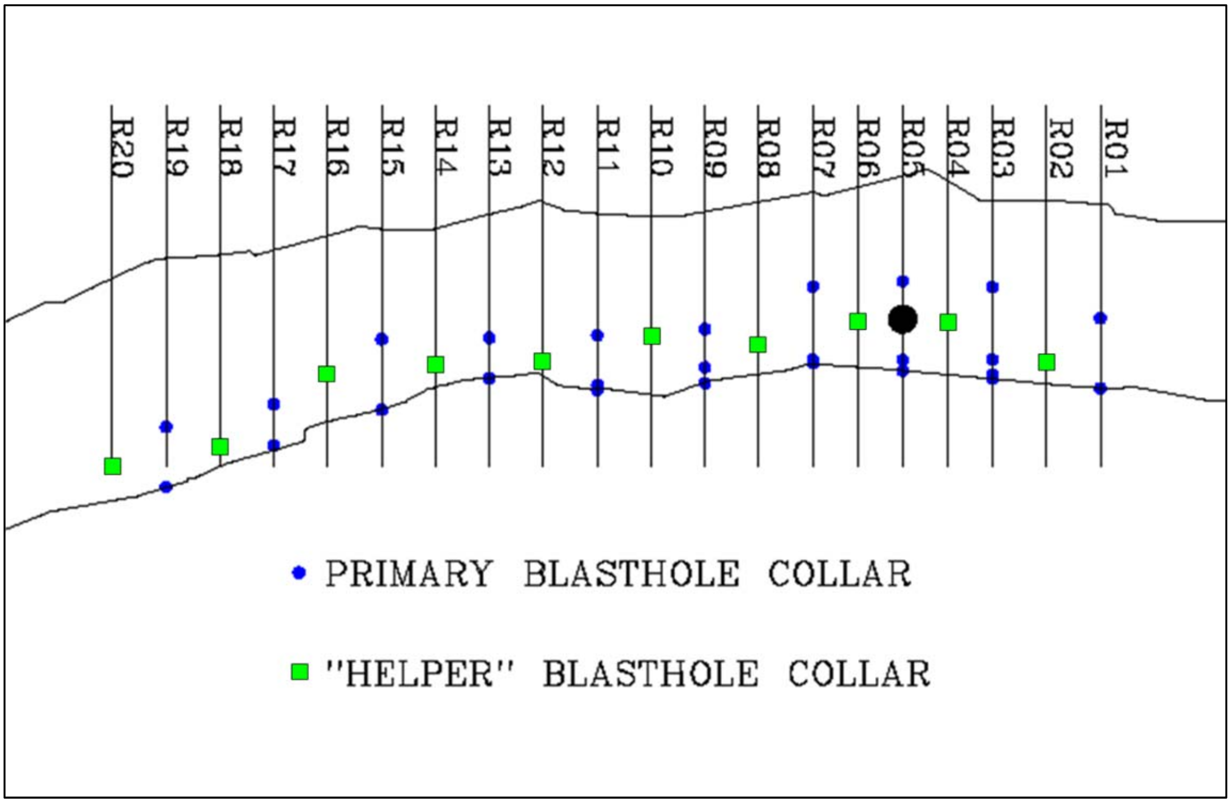


Figure 7-2 Plan view illustrating the dice-five/dice-seven drilling pattern layout.

7.2 Stope Geometry and Dilution Assessments

Stope analysis and design are completed on the surfaces of the open stopes. The geometry of a stope surface is quantified based on the stope hydraulic radius (Section 3.2.2) and the inclination angle of the hanging wall. These terms are required when using existing empirical charts for predicting whether a stope surface will remain stable, and for determining the quantity of dilution associated with a given stope surface. Figure 7-3 shows a typical stope cross section at the Eagle Point Mine. This includes the stope overcut and undercut drifts, the planned stope blast outline, projected hanging wall geological structures, the location of cable bolts, and the cavity monitoring survey (CMS, as described in Section 3.4) (Forster et al., 2007). This information is generally available for every stope, and each component of the stope information contributes to the stope reconciliation process.

Although there are several methods of measuring the stope geometry, it is most important to ensure that the method of measurement is standardized and repeatable for a particular mine site. One of the objectives of this project is to develop a methodology for stope reconciliation, including stope geometry. There are many possible approaches for measuring stope geometry and they are discussed in this section.

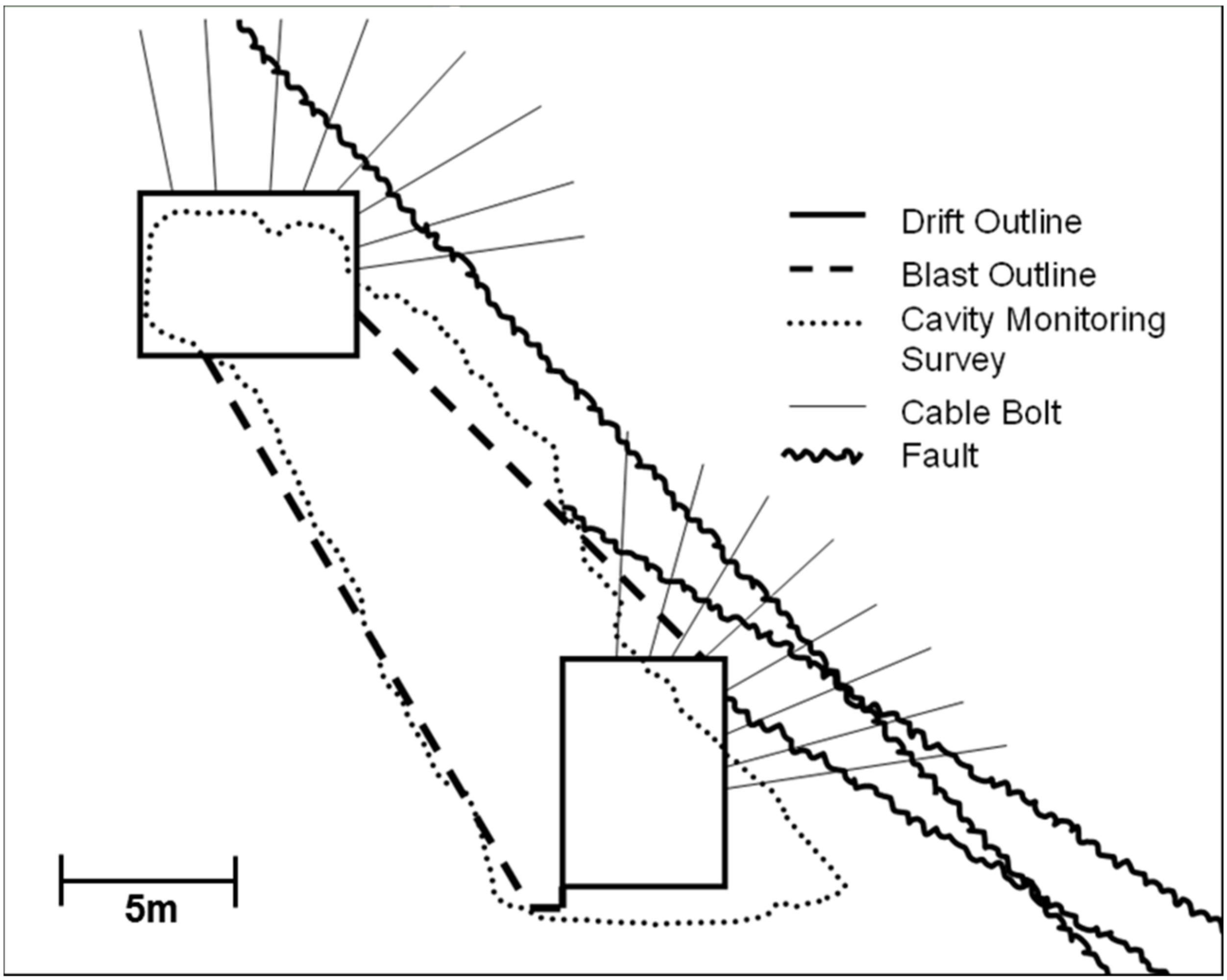


Figure 7-3 Typical cross section of a stope showing the drift outlines, planned blast outline, cable bolts, results of the cavity monitoring survey (CMS) and geological structures (From Forster et al., 2007).

7.2.1 **Stope Up-Dip Extent**

There are several measurements which are possible for the up-dip length of a stope. Figure 7-4 shows three proposed up-dip stope lengths, which include the cable supported overcut and undercut drifts as shown. The first proposes using the length from the top of the overcut to the bottom of the undercut, ignoring the effect of the cable support in the overcut and undercut drifts. The second uses the length along the hanging wall side of the blast outline, ignoring the overcut and undercut drifts by treating the cable bolt support as a rock abutment. The third approach uses the length from the mid-point of the overcut to the mid-point of the undercut.

For larger stope sizes, the choice of the up-dip length has less influence on the calculated hydraulic radius because the overcut and undercut drifts make up a smaller percentage of the total up dip length. For the relatively small stope sizes common at the Eagle Point Mine, the choice of up-dip strike length has a much greater effect on the overall hydraulic radius.

For this project, the projected length along the hanging wall drill hole from the center of the overcut to the center of the undercut was used for the hydraulic radius calculation. If the hanging wall drillhole did not intersect the centre of either the overcut or undercut drift, the centre point of the drift was projected horizontally. The intersection of the projected hanging wall drillhole to the horizontal line was used for the up-dip hanging wall length. Cable bolt support in both the overcut and undercut drifts at the mine is extensive, and is usually quite effective. To include the entire height of the undercut and overcut drifts for the hydraulic radius calculation would be overly conservative. Conversely, to exclude the entire height of the overcut and undercut drifts for the hydraulic radius calculation would be under-conservative. Sloughing and failure often extend into the cable bolted zones and cannot be compared to the support provided by the rock abutments. Measuring to the center of the overcut and undercut drifts is easily repeatable, and this method assumes that the ground support provides some influence on the stope stability.

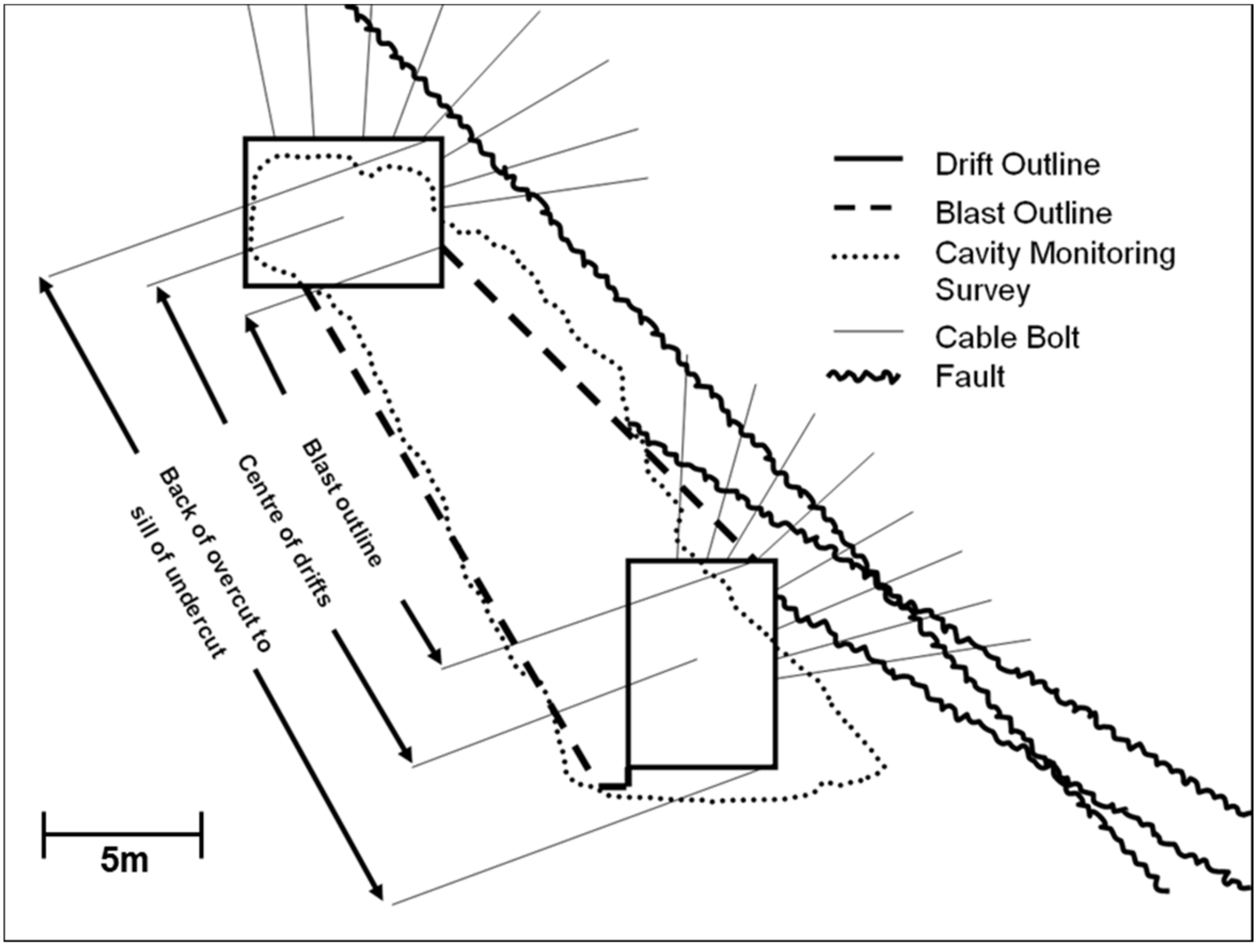


Figure 7-4 Three potential measurements of the up-dip hanging wall length which may be used to calculate the hydraulic radius of a stope (From Forster et al., 2007).

7.2.2 **Stope Inclination**

There are different options for defining the inclination angle of a stope. As introduced in Section 3.2.1, the stability number, N', includes the stope orientation factor, C (Potvin, 1988). This factor is based upon the angle of the hanging wall. Figure 7-5 shows two of the possible angles which may be used to calculate the C factor (Forster et al., 2007). One angle is the hanging wall blast outline angle, while the second is the angle from the top hanging wall corner of the overcut to the bottom hanging wall corner of the undercut.

For this study a third approach was taken. The average angle of the primary ring hanging wall drill holes, as illustrated in Figure 7-1 and Figure 7-2, was used for the calculation of the modified stability number (N') gravity reduction (C) factor. These hanging wall drill holes will have the greatest influence on the overall stope hanging wall inclination and may vary by over 30° due to the complexity of the orebody. The measurements are also repeatable for each stope.

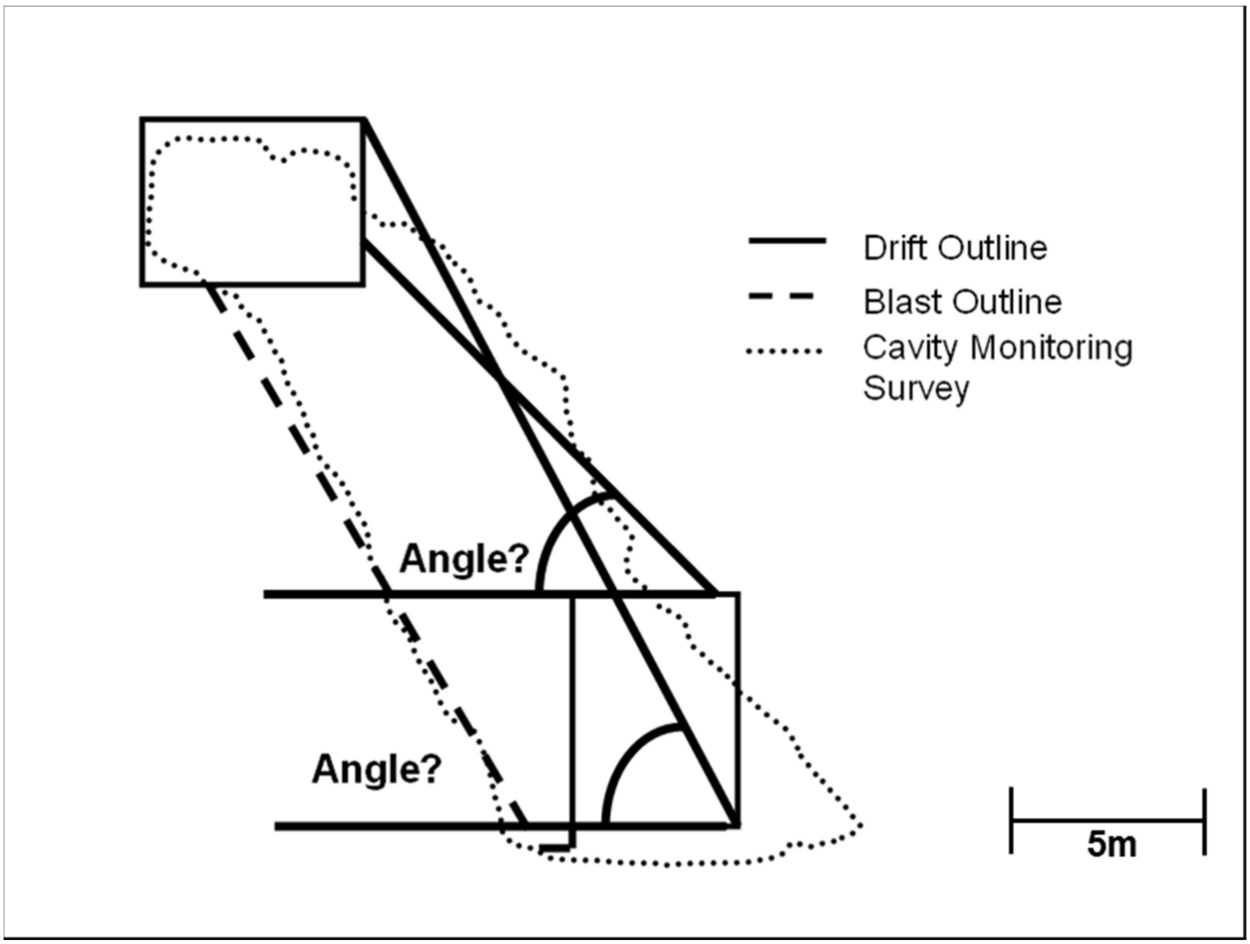


Figure 7-5 Cross section depicting two of the angles which potentially be used to calculate the gravity reduction factor, C (From Forster et al., 2007).

7.2.3 **Stope Hanging Wall Undulation**

A further complication of the stope reconciliation process is the three dimensional variations in the stope shapes. The underground tunnels are developed following indications of the mineralization. This most commonly results in an undulating shape for the drift. Using data from a single two-dimensional cross section of the stope may lead to over- or underestimation of the dilution for the stope.

Cross sections through the stope will show significant changes in both the ore body dip and also the up-dip extent of the ore body. This variability influences both the hanging wall gravity reduction factor C, based on the hanging wall dip, and the hanging wall up dip length and corresponding hydraulic radius.

7.2.4 **Overbreak / Dilution Assessment**

Commonly, the stope hanging wall geometry for stope reconciliation is chosen to coincide with the ore contact. It is assumed that the blast holes will be stepped back from the contact to allow the blast to break to the contact. At the Eagle Point Mine, the blast pattern varies significantly with rock mass conditions, undercut and overcut location, and the blast engineer's judgment. For consistency, it is necessary to take the primary ring blast hole furthest into the hanging wall as the contact as the datum from which dilution is measured.

7.2.5 **Stope Reconciliation Methodology**

Figure 7-6 shows a typical example of a stope with undulating drifts, and varied dilution along the hanging wall. Figure 7-7 is a section view through the case study presented in Figure 7-6. The figure shows the stope overcut and undercut, the cable bolt support installed in both drifts, the geological ore model, local geological structures, the production drill holes, and the cavity monitoring survey results.

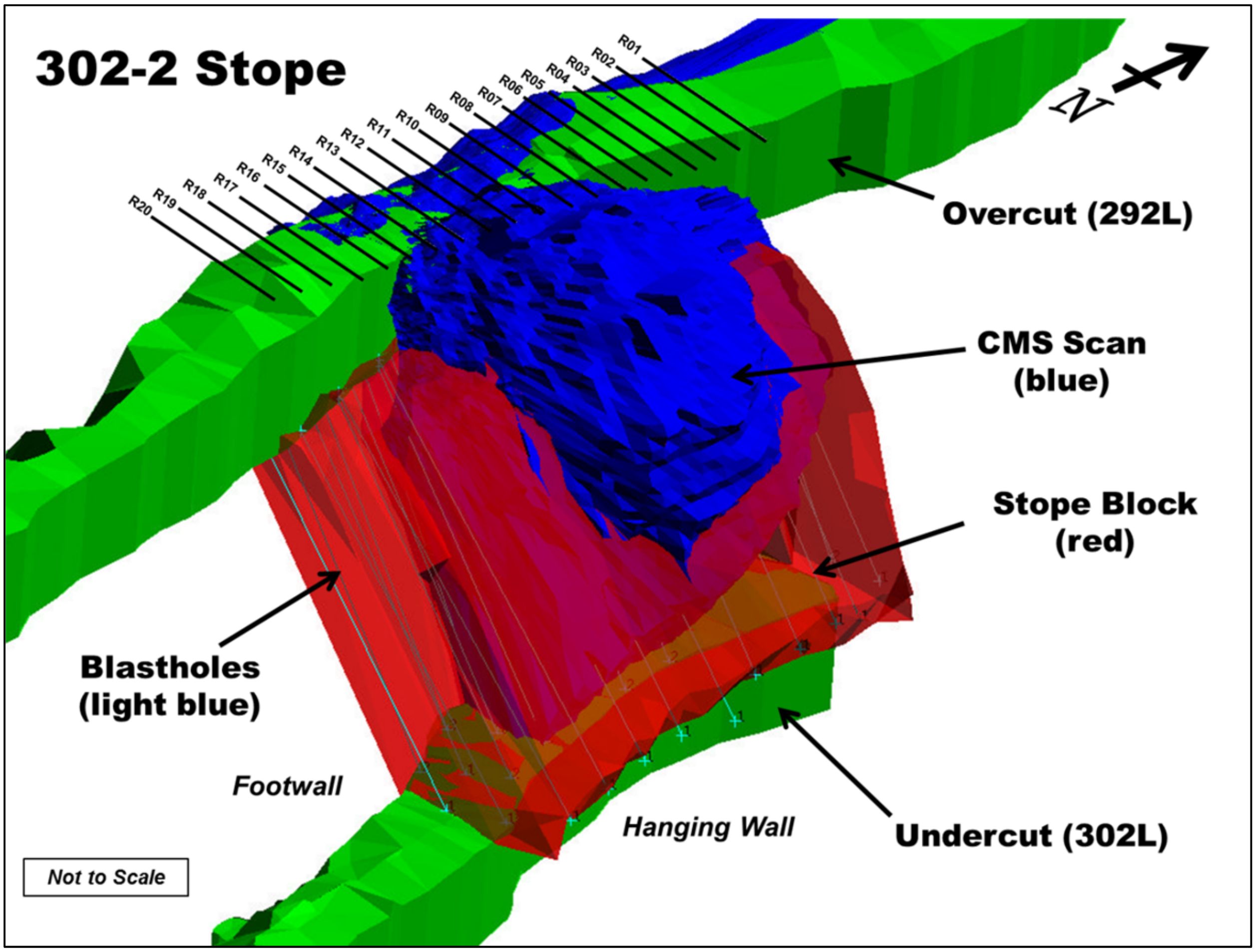


Figure 7-6 Isometric view of stope 302-2 depicting the overcut and undercut drifts, the stope outline, the results of the cavity monitoring survey, and the designed blastholes (After Forster, 2011).

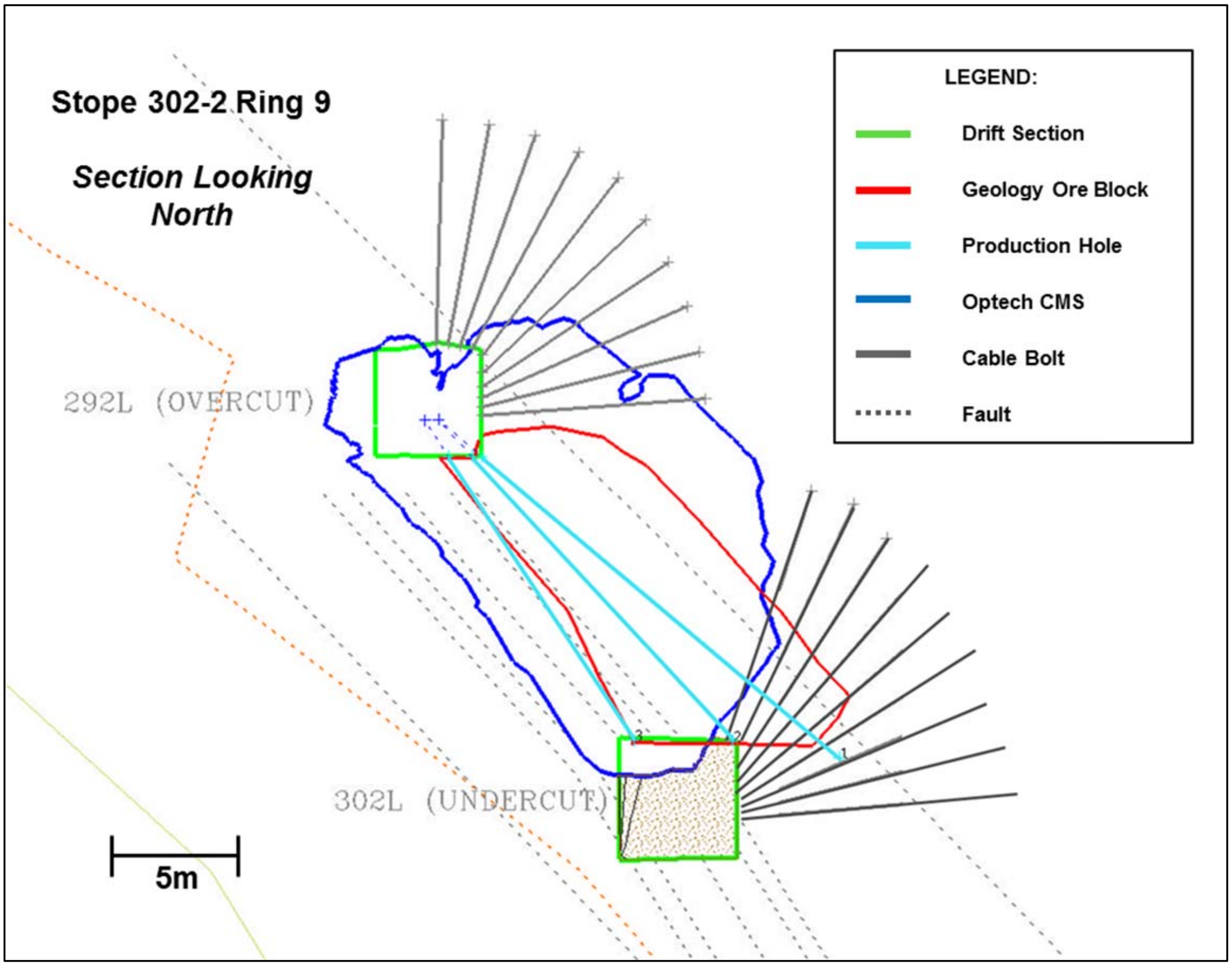


Figure 7-7 Section view through stope illustrating the drift sections, cable bolt support, production drill holes, geological ore block, geological structures, and cavity monitoring survey outline (After Forster, 2011).

Figure 7-8 summarizes the standard reconciliation measurements that were done for each primary blast ring. The up-dip hanging wall length is the length of the projected hanging wall drill hole to the centroid of the overcut and the centroid of the undercut. The overall hanging wall up-dip length to be used for the hydraulic radius calculation is the average of the lengths measured from all of the sections. The hanging wall angle is the angle of the hanging wall drill hole. The overall hanging wall angle to be used for the "C" factor of the N' value is the average of the angles measured from all of the primary ring sections. The area for the ELOS calculation is the area between the cavity monitoring survey outline and the hanging wall drill hole. To assume a volume of overbreak, each ELOS is multiplied by the distance to adjacent "helper" rings. The overall volume for the ELOS calculation is an average of the volume values for each of the primary sections. The dilution is the average volume divided by the surface area of the hanging wall to give an average depth of failure, or ELOS.

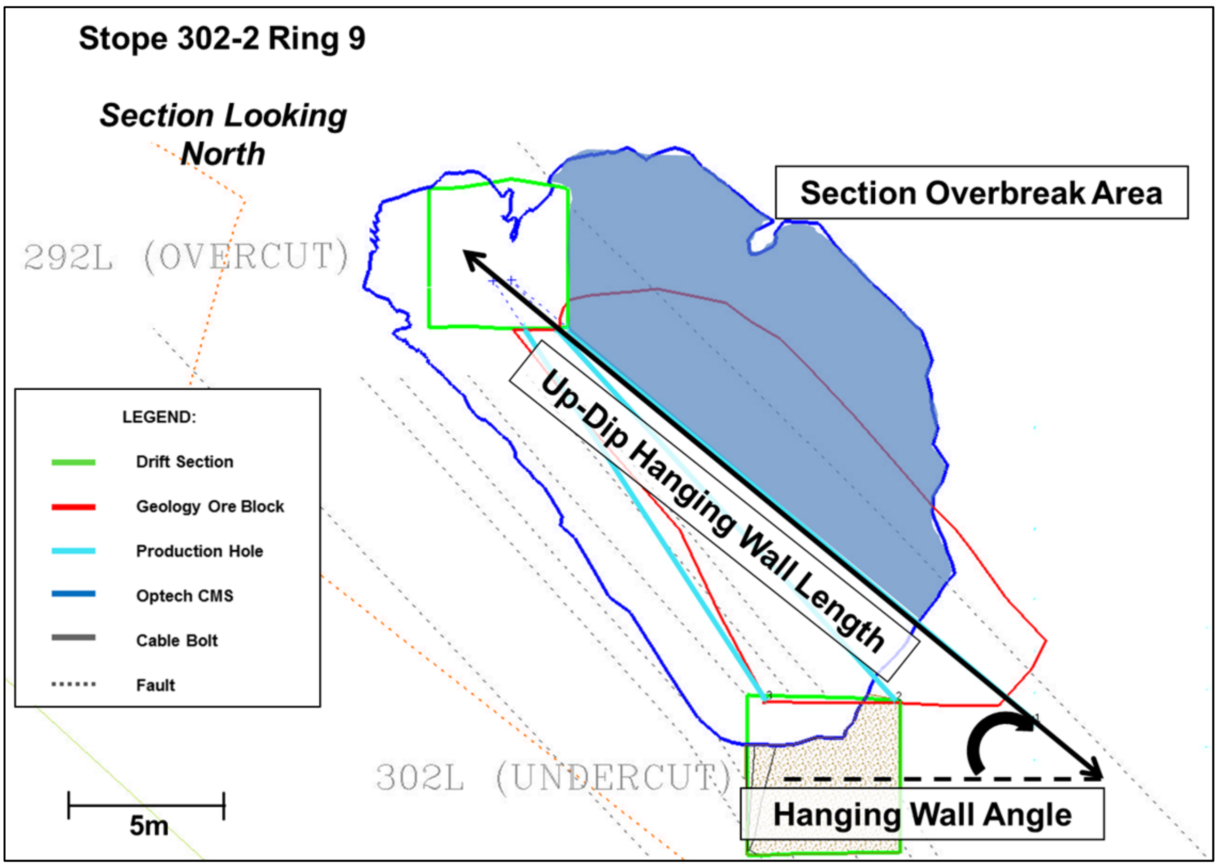


Figure 7-8 Section view illustrating the common measurements for stope reconciliation sections (After Forster, 2011).

The modified stability number, N', as well as factors A, B, and C are also considered. For the A factor, Potvin (1988) conducted a modelling parametric study and proposed that stresses in the hanging wall will be low if the up dip and along strike dimensions of the hanging wall are much greater than the stope width. Additionally, prevalent fault and shear zones within the rock mass will act to prevent high stresses, as these zones will tend to deform and shed high stresses. As such, the A factor is assumed to be 1.0 for all stopes. This can be represented by the red line in Figure 7-9, corresponding to induced stresses less than 10% of the intact rock strength.

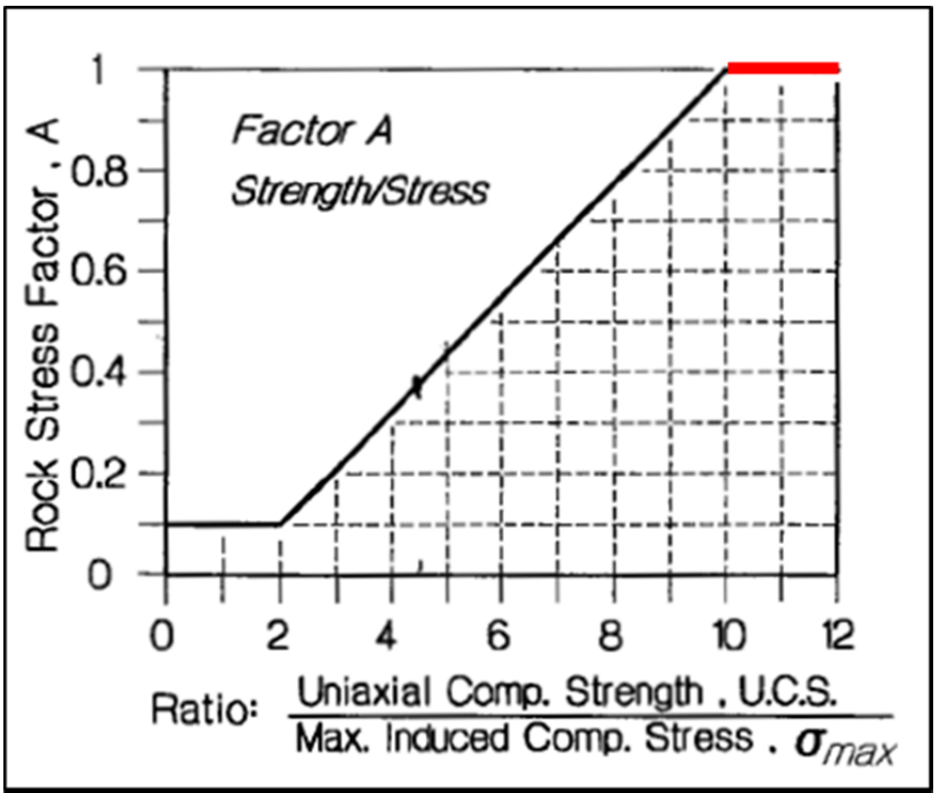


Figure 7-9 Rock Stress Factor, A, for Stability Graph Analysis (After Potvin, 1988, from Hutchinson and Diederichs, 1996). Red line indicates A = 1.0 for stopes in relaxation or low stress conditions.

The B factor is based on the relative difference between the surface being analyzed, and the predominant joint set. In the case of the stope hanging wall, the joint set to be analyzed is the one occurring approximately parallel to the hanging wall (Joint set A, Section 2.1.2). An example of the structural mapping dip and dip directions are shown in Figure 7-10. These measurements were entered into the Rocscience graphical and statistical analysis of orientation data program, Dips™.

Contour plots of the dip and dip direction measurements from the back mapping information are shown for 292L (Figure 7-11 a) and for 302L (Figure 7-11 b). Because the results were only obtained from the back mapping, any joint sets occurring parallel or sub-parallel to the back are unlikely to be recorded in the back mapping. As such, the contour plots in Figure 7-11 are not representative of the number of joint sets, but can be used to determine the average dip and dip direction of the joint set most similar to the stope hanging wall. Joint set A is near parallel to the hanging wall, however, due to the irregular nature of the ore contact, a 0 to 20° variation often occurs, resulting in a B value of 0.2. A B factor of 0.2 (Figure 7-12) is used for all stopes as it is most reflective of actual conditions, and is the most conservative value to be used for stability assessments.

The C factor is calculated on a stope-by-stope basis using Equation 3.2. The C factor may be any value from 2 to 8 depending on the average stope hanging wall angle, as shown by the red line in Figure 7-13.

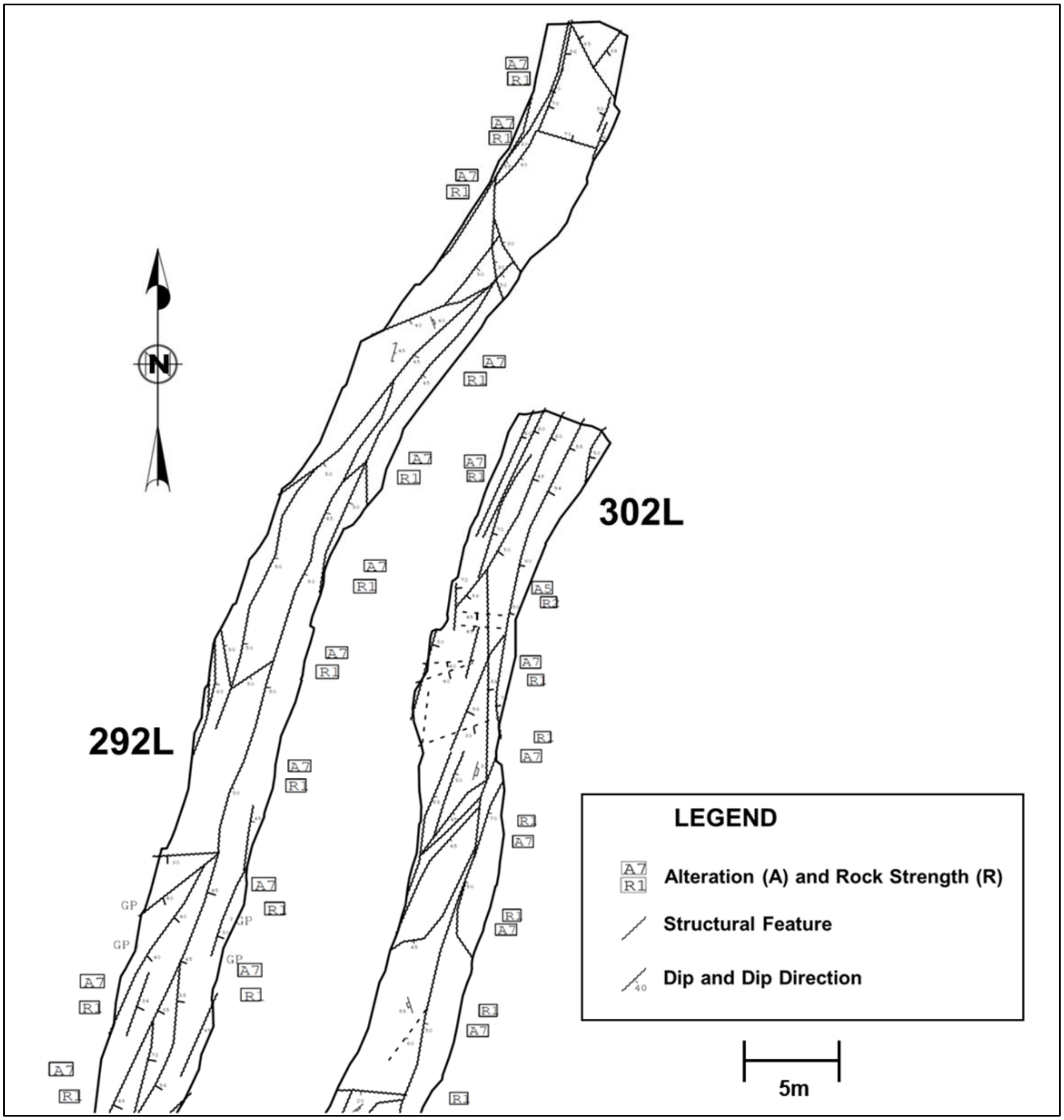


Figure 7-10 Structural mapping information for 292L and 302L.

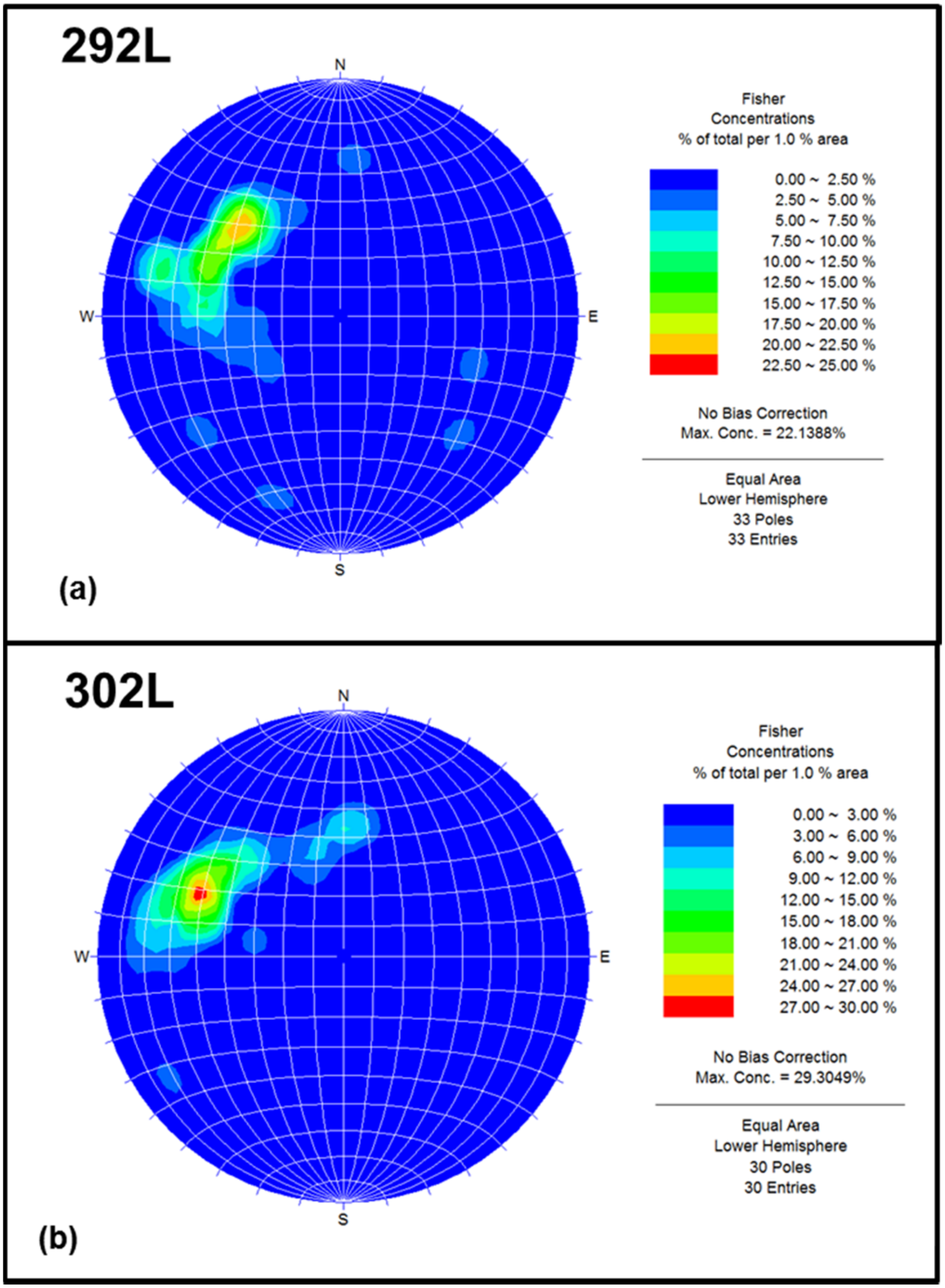


Figure 7-11 Equal area contour plots of dip and dip direction measurements from back mapping for 292L (a) and 302L (b).

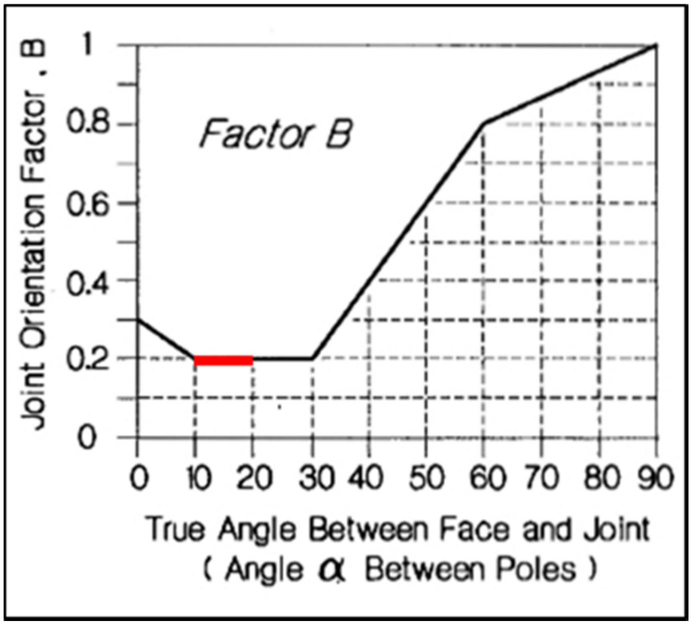


Figure 7-12 Determination of Joint Orientation Factor, B, for Stability Graph analysis (After Potvin, 1988, from Hutchinson and Diederichs, 1996). Red line indicates B = 0.2 for true angle between face and joint of 10° to 20°.

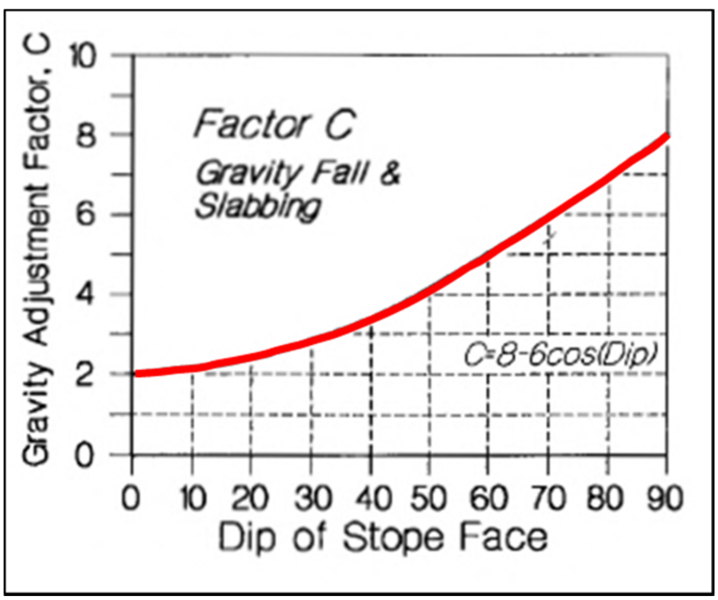


Figure 7-13 Determination of Gravity Adjustment Factor, C, for Stability Graph analysis (After Potvin, 1988, from Hutchinson and Diederichs, 1996). Red line indicates C = 2 to 8, depending on the average stope hanging wall angle.

The average ELOS, as measured from the cavity monitoring survey data, is used to compare the predicted ELOS to the results after mining. It is important to note that the economic dilution, or un-mineralized rock material removed with the ore material, will not necessarily be the same as the geotechnical dilution, or the more conventional ELOS definition as introduced in Sections 1.2 and 3.3.1. The economic dilution is the waste material that exists outside of the geological stope blocks, and a certain amount of overbreak will be planned with the production drill hole and blast design. The geotechnical dilution definition for this project is all of the material that is removed outside of the hanging wall drill hole, and essentially ignores the geological block model in the analysis.

7.3 Stope Rock Classification Assessment

As discussed, the ore body and hanging wall geology are highly complex and the sources of data for assessing the rock mass condition are varied. The most complete data set for characterizing the rock mass are the geology A/R ratings for the overcut and the undercut. Additional data is also available from the exploration diamond drill holes used to help delineate the mining and stoping limits. Figure 7-14 is a plan view of the geological mapping and classification of an example stope overcut. The stope extents analyzed and the cavity monitoring survey data covered the total extent of the mined stope. Two zones of rock type and classification were identified, as shown with the blue lines in Figure 7-14. The rock type zones were analyzed and measured along the hanging wall side of the drift. One rock type zone is A7/R1 biotite-quartz-feldspar gneiss, and the other zone in this example is A7/R1 pegmatite. Similar measurements were made for the undercut drift.

In addition to the overcut and undercut drift mapping data, RQD data is available from diamond drill hole core used to help delineate the ore body extent. As many of the exploration holes are oriented normal to the stope hanging wall, the RQD information from the diamond drill holes is very useful. However, the relatively small size of the stopes may result in very few exploration holes intersecting the stope.

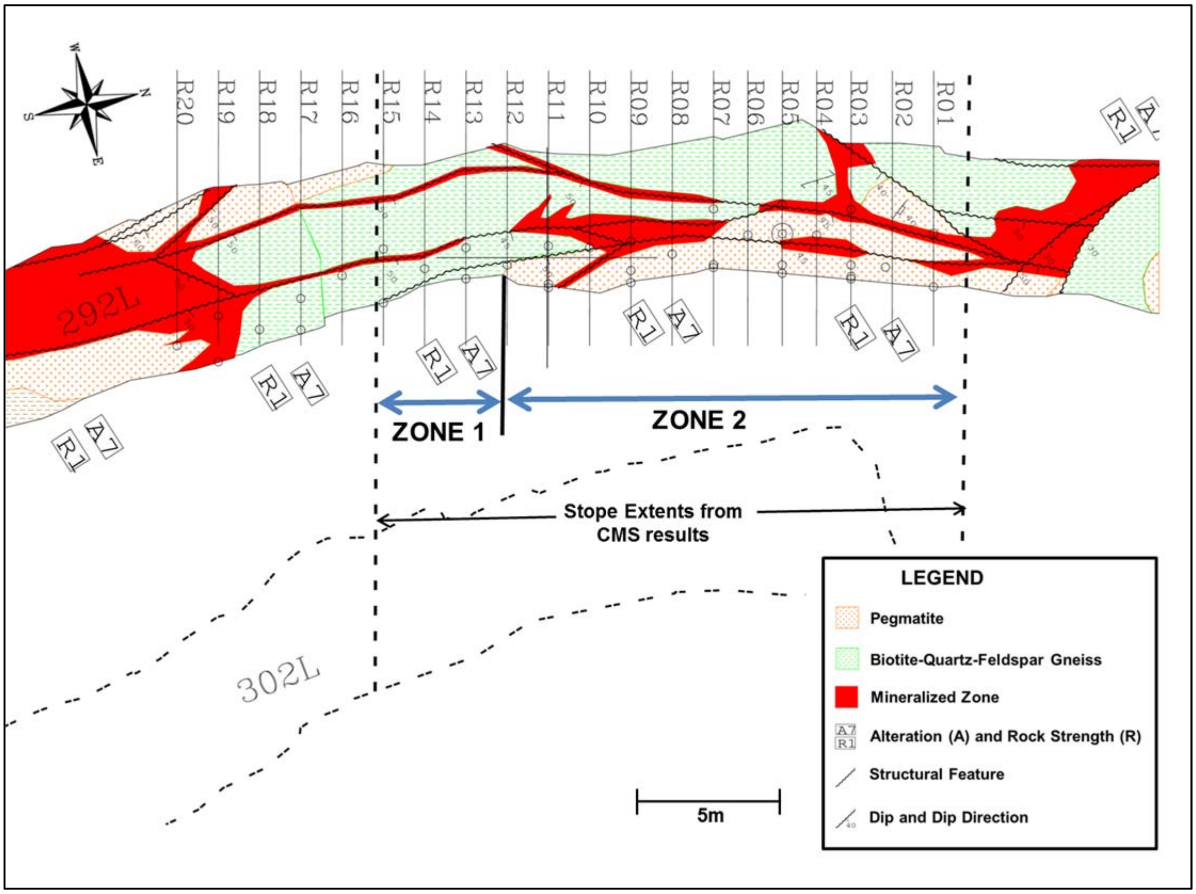


Figure 7-14 Plan view of an overcut drift with the geological mapping and A/R values shown.

To augment the data used to estimate RQD, both the estimated RQD value from the geology A/R drift mapping and the RQD value from the core logging are included in an estimate of RQD. It should be noted that the RQD measurements are not part of the A/R assessment from drift mapping. The RQD from drift mapping is based on the correlation between A/R characterization and the RQD discussed in section 6.1.2. Part of the initial planning process is to assess the quality of the rock mass. Since the stope block and blast hole layout are often not available during the stope planning and geotechnical analysis, the stope limit for RQD assessment is taken as the first core run on the hanging wall side of the ore contact as assessed by the site geologists. This may not correspond with the hanging wall limit based on the blast hole layout, as shown in Figure 7-15, but it is operationally more manageable for developing a consistent methodology for stope reconciliation and dilution prediction. Figure 7-15 shows a cross section of a stope.

The red outline is the stope ore block, and there is a diamond drill hole intersecting the hanging wall of the stope which has the RQD values annotated, corresponding to approximately 3 metre core runs. In this example, the RQD value for this drill hole which would be used in the reconciliation would be 30%.

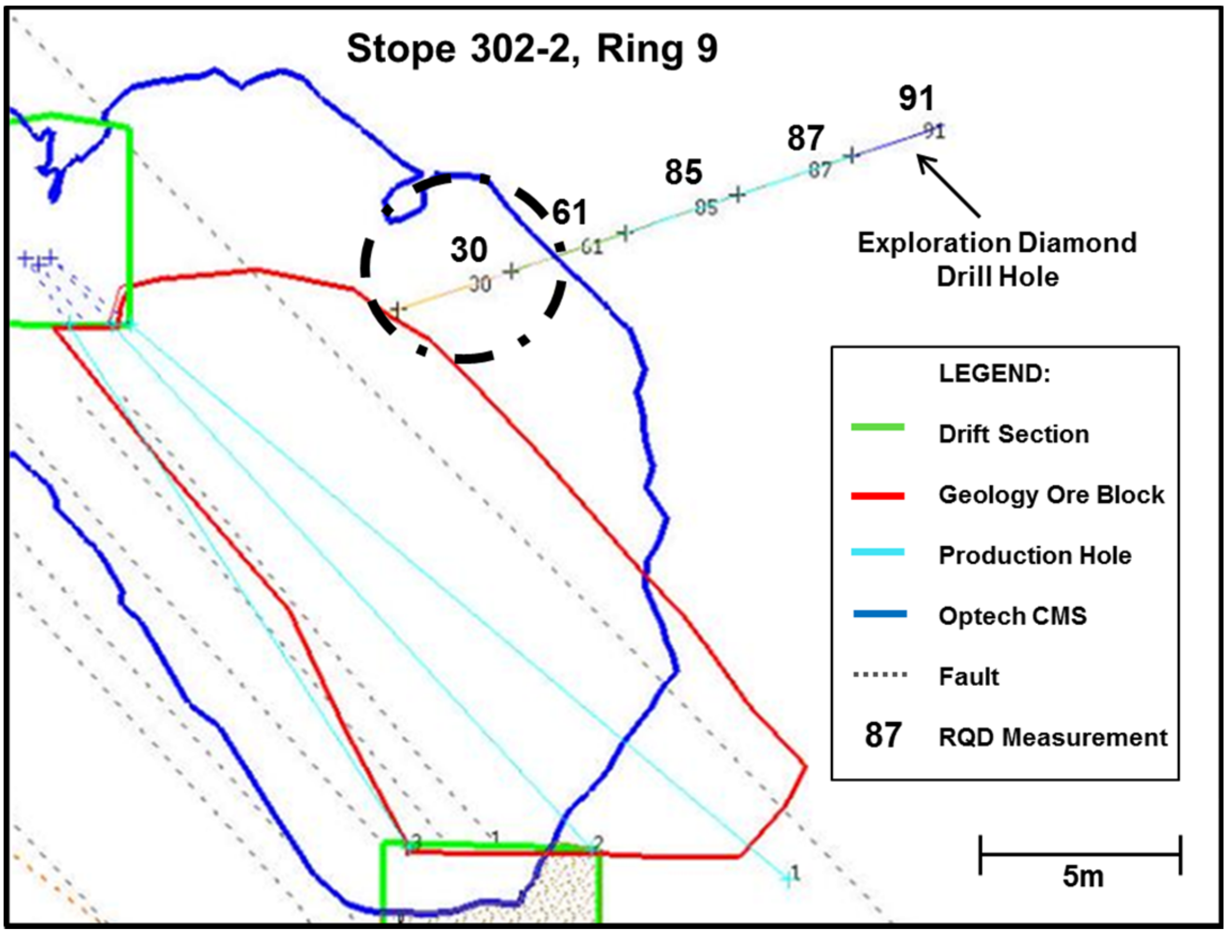


Figure 7-15 Section view of a stope with a diamond drill hole intersecting the stope from the hanging wall side. Exploration diamond drill hole RQD values are shown. (From Forster, 2011)

The RQD measurements from the exploration diamond drill holes have an impact on the overall RQD value for the entire hanging wall of a stope, but the hanging wall area coverage that each diamond drill hole corresponds to is limited. The cross-sectional area of each drill hole is approximately 0.01 m^2. With the highly variable lithology and structural geology, it is difficult to assess how much area of the hanging wall can be represented by the RQD measured in a drill hole. Assuming a stope strike length of 20 metres and a hanging wall up-dip length of 20 metres, the hanging wall area would be 400 m^2. The drift locations may not be in the ideal location for the stope block and may be more representative of the stope block than the hanging wall area. The maximum hanging wall area that may be represented by each drift would be 80 m^2, assuming a drift height of 4 metres and a strike length of 20 metres, but it may be significantly less. Although the diamond drill hole RQD values would be most representative of actual hanging wall conditions, the area which they would influence is highly variable. It was necessary to use both the drift and diamond drill hole estimate of RQD because, in some cases, there may be no diamond drill holes that intersect the stope. The drift mapping RQD values are more representative of a larger area, but may not reflect actual hanging wall conditions. A combination of the sources is required.

A simple, easily followed approach for obtaining an average RQD estimate for the stope hanging wall is proposed. The drift mapping RQD estimate is based on the A/R to RQD approximation shown in Figure 6-5 and Figure 6-6 (Section 6.1.2). The RQD values for the drifts are based on variable correlations, not from direct measurements.

The length of each rock type and classification zone is measured for the stope overcut drift. The length measurements are limited to the stope extents. The corresponding Q’ parameter value for each rock type and classification zone is multiplied by the length of hanging wall exposure. The sum of all of the zone measurements is divided by the total drift length to determine a weighted parameter rating for each drift is shown in Equation 7.1.

$$P_{drift} = \frac{\left[\left(P_{Ax/Rx} \times x\right) + \left(P_{Ay/Ry} \times y\right) + \left(P_{Az/Rz} \times z\right) + \cdots\right]}{(x + y + z + \cdots)}$$

(Equation 7.1)

where,

P_{drift} = RQD, J_n, J_r, or J_a parameter for the overcut or undercut drift

A(x,y,z,…)/R(x,y,z,…) = geological mapping alteration and strength category value by rock type

x, y, z,… = length measurement for geological alteration and strength category value by rock type

The $RQD_{average}$ is the arithmetic mean of the overcut RQD, the undercut RQD, and the number of RQD values from the diamond drill holes.

$$RQD_{average} = \frac{(RQD_{overcut} + RQD_{undercut} + RQD_{DDH\,1} + RQD_{DDH\,2} + \cdots)}{(2 + Number\ of\ Drillholes)}$$

(Equation 7.2)

The $RQD_{average}$ value incorporates the diamond drill hole information in such a way that as the number of drill holes intersecting the stope increases, the influence of the assumed RQD values from the geological A/R mapping decreases.

In order to apply the A/R geology assessment and RQD measurements for stope design and dilution prediction, a representative estimate of rock quality, Q', is required.

The weighted actual Q' is essentially the average of the Q' of the overcut and undercut, and also includes RQD values from diamond drill hole information. The weighted actual Q' value is calculated using the following formula:

$$Q'_{weighted} = \frac{RQD_{average}}{\left(\frac{\sum J_{n(A/R)}}{2\,l}\right)} \times \frac{\left(\frac{\sum J_{r(A/R)}}{2\,l}\right)}{\left(\frac{\sum J_{a(A/R)}}{2\,l}\right)}$$

(Equation 7.3)

where,

$J_{r(A/R)}$ = (J_r value from rock type and A/R) x (Hanging wall length of the rock type and A/R)

$J_{n(A/R)}$ = (J_n value from rock type and A/R) x (Hanging wall length of the rock type and A/R)

$J_{a(A/R)}$ = (J_a value from rock type and A/R) x (Hanging wall length of the rock type and A/R)

l = Strike Length

$RQD_{average}$ = arithmetic mean RQD value (Equation 7.2)

The $Q'_{weighted}$ value, the arithmetic mean of the up-dip stope length, and the arithmetic mean of the hanging wall angle are all components of the dilution prediction analysis using the Modified Dilution Graph as shown in Figure 3-9.

Three approaches are presented for the interpretation of the available data for an estimation of Q'. The first approach takes the A/R and rock type data and correlates it to a Q' value based on work conducted by Sutton (1998) and Sutton and Milne (1998). As mentioned in Section 5.4.1, this correlation is based on very few observations. The second approach takes the RMR_{76} data discussed in Section 5.4.2, and links it to Q' based on Equation 2.4.

The third approach uses the RMR_{76} observations, Sutton's data and mapping conducted as part of this research project and correlates the data to each parameter required in the Q' classification, as discussed in Section 6.2.1.

7.3.1 **Sutton Q' Approach**

The mine has historically used the Sutton Q' from geology A/R values (Table 5-4) for cable bolt design and stope dilution prediction. Some of the design engineers would arbitrarily choose an average geological A/R value to represent the stope. The diamond drill holes would be evaluated, and lower RQD values would subjectively influence the engineer to a lower value within the range observed in the A/R correlation. Other design engineers would choose individual Q' parameters based on their personal experience and observations, and subjective values for the RQD parameter based on the diamond drill holes. Although neither of these approaches is inherently wrong, the subjectivity in the method makes the comparison of results and repeatability of the method extremely difficult.

One objective of this project was to define a standardized method of evaluating the stope geometry; another objective was to improve the geology A/R classification to Q' correlation. As such, the method of estimating a Q' value for a stope as presented in Equation 7.3 will be used to compare the Sutton Q' values and the Parameter Based Q' values. The RMR_{76} to Q' equation based approach is discussed in the following section.

7.3.2 **RMR_{76} to Q' Equation Based Conversion Approach**

Average RMR_{76} values were calculated for each rock type as shown in Table 7-1. These RMR_{76} values were converted to Q' values using Equation 2.4 as shown in Table 6-6. The average RQD and RMR_{76} values by rock type and A/R geology classification are shown in Table 7-1. As well as using the conversion of RMR_{76} to Q' using Equation 2.4, the RQD values were calculated separately from the RMR_{76} mapping data to allow separate averaging of RQD. No RMR_{76} observations for the A3/R2 pegmatoidal rock type were found, so the RQD and RMR_{76} values for it were interpolated as a value between the A5/R2 and A5/R3 pegmatoidal observations.

Table 7-1 Average RQD values and RMR_{76} observations by rock type and A/R geology classification.

Rock Type	A/R Classification	Average $RQD_{RMR\ 76}$	Average RMR_{76}	No. of Ratings
Pegmatoidal	A7/R1	38	34.0	2
	A5/R1	25	32.0	1
	A7/R2	73	58.0	2
	A5/R2	63	52.1	14
	A3/R2	71**	60**	0**
	A5/R3	79	70.0	2
	A3/R3	62	60.4	5
	A1/R3	73	68.5	2
Gneissic	A7/R1	69	55.5	6
	A5/R1	38	41.0	1
	A7/R2	73	58.5	2
	A5/R2	68	63.0	14
	A3/R2	78	64.3	4
	A5/R3	73	67.8	4
	A3/R3	77	68.8	10
	A1/R3	63	61.0	1

***No exposures were mapped. Values inferred.*

The method of using the RMR_{76} to Q' ratings for stope dilution prediction is similar to the method used for the Sutton Q' approach, with one key difference. Section 6.1 discussed the parameters that the RMR_{76} and Q systems have in common, and illustrated the parameters that are not directly equivalent. The RMR_{76} values cannot be separated into individual Q' parameters using Equation 2.4. In order to ensure that the dilution predictions are assessed equally to compare the rock mass classification input values, the diamond drill hole RQD assessments must also be included. The RQD observations from the RMR_{76} site mapping, averaged for the A/R classification (Table 7-1) are one of the parameters that can be directly compared for the RMR_{76} and Q' classification systems. The weighted Q' value ($Q'_{weighted}$) was calculated using the following formula:

$$Q'_{weighted} = \left(\frac{Q_{weighted\ o/c} + Q_{weighted\ u/c}}{2}\right) \times \frac{\left(RQD_{average}\right)}{\left(\frac{RQD_{weighted\ o/c} + RQD_{weighted\ u/c}}{2}\right)}$$

(Equation 7.4)

where,

$Q'_{weighted\ o/c}$ = (Q' value from rock type and A/R) x (Hanging wall length of the type and A/R) for the overcut drift

$Q'_{weighted\ u/c}$ = (Q' value from rock type and A/R) x (Hanging wall length of the type and A/R) for the undercut drift

$RQD_{weighted\ o/c}$ = (RQD value from rock type and A/R) x (Hanging wall length of the type and A/R) for the overcut drift

$RQD_{weighted\ u/c}$ = (RQD value from rock type and A/R) x (Hanging wall length of the type and A/R) for the undercut drift

$RQD_{average}$ = arithmetic mean RQD value (Equation 7.2)

The use of the RMR_{76} to Q' equation based conversion approach will have no impact on the A, B, and C factors used to calculate the modified stability number, N' as the measurements for these factors are independent of the rock type.

7.3.3 **Example Stope 302-2 Case Study**

To illustrate the stope reconciliation process used for this study, the measurements and results of the 302-2 stope are presented. Figure 7-16 shows a plan view of the stope overcut level (292L) with the stope production rings, the geological mapping information, and the alteration/strength (A/R) values. Also shown are the total stope length that was measured using the stope cavity monitoring survey results, and the length of the zones of the rock types and A/R values. For the example shown, the surveyed stope strike length was measured as 21.5 m. There are two rock mass classification zones; 4.5 m of A7/R1 biotite-quartz-feldspar gneiss and 17.0 m of A7/R1 pegmatite in the stope overcut.

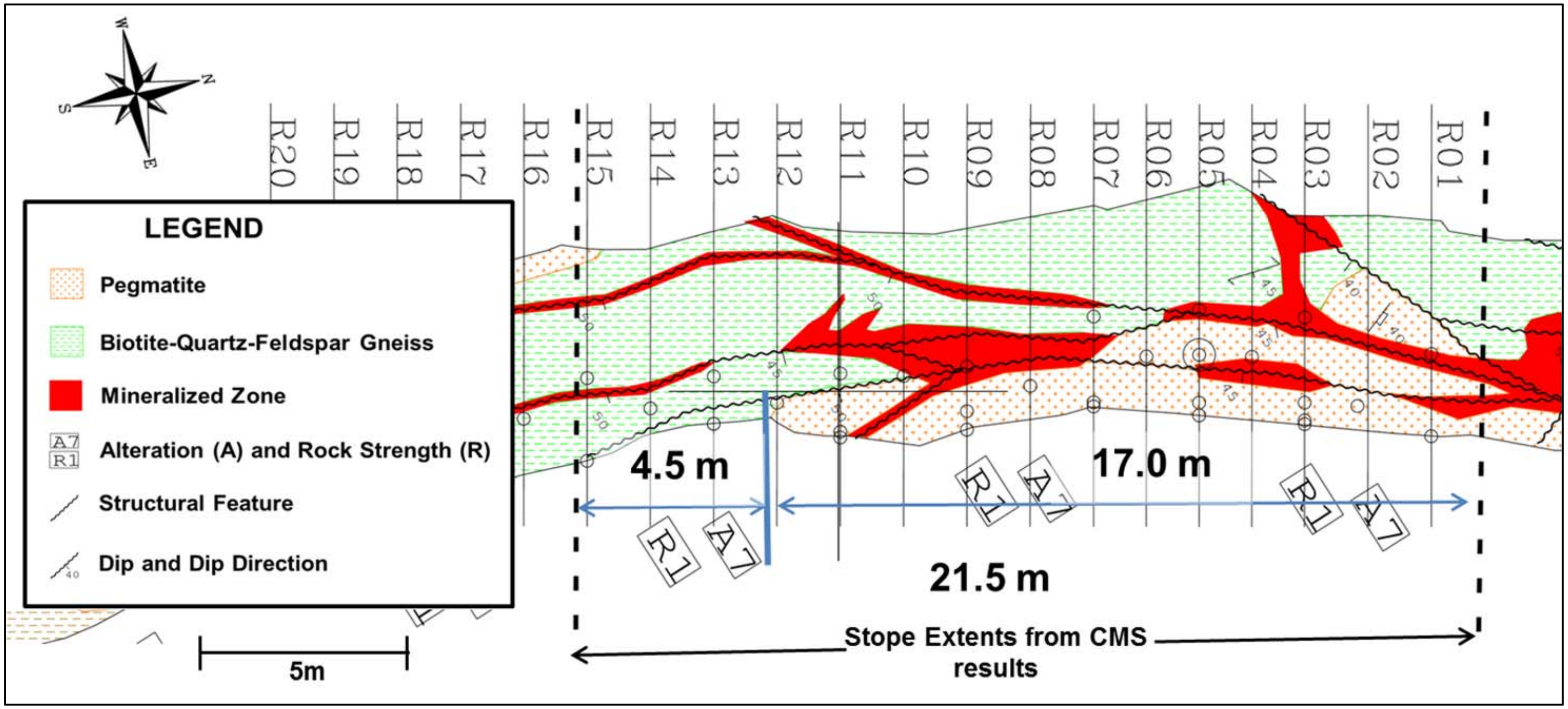


Figure 7-16 Geological mapping, stope rings, and A/R extents for 292L (302-2 stope overcut).

Table 7-2 shows the correlated RQD, J_n, J_r, and J_a values and their weighting based on exposure extent for the Sutton Q' Approach and Parameter Based Q' Approach, and the correlated RQD and Q' values and their weighting based on exposure extent for the RMR_{76} to Q' Equation Based Conversion Approach for the stope overcut. Table 6-4 was used for the Sutton Q' Approach and Parameter Based Q' Approach for each input parameter for the corresponding rock type and A/R category. The individual parameters were multiplied by the length of the rock type and A/R category. Table 7-1 and Equation 7.3 were used for the RMR_{76} to Q' Equation Based Conversion Approach for each input parameter for the corresponding rock type and A/R category. The RQD and Q' parameters were multiplied by the length of the rock type and A/R category. The multiplication values were summed and then divided by the strike length to get a weighted value for each parameter.

A similar process was completed for the undercut of the stope (302L), and the results are shown in Figure 7-17 and Table 7-3. It should be noted that the undercut of the stope was not developed to the end of the stope block. There is a 2.0 m section of the undercut of the stope where there is no information on the rock type or A/R category. The strike length of the undercut was altered to accommodate the difference. Photographs of the stope overcut and undercut near the centre of the stope are shown in Figure 7-18, Figure 7-19, and Figure 7-20 to illustrate the quality of the rock mass.

Table 7-2 Calculations of the weighted RQD, J_n, J_r, and J_a values for the Sutton Q' Approach and Parameter Based Q' Approach, and the weighted RQD and Q' values for the RMR_{76} to Q' Equation Based Conversion Approach for the 302-2 stope overcut (292L)

Overcut (292L) - Sutton Q' Approach										
Rock Type	**A/R value**	**Length (m)**	**RQD**	**RQD x L**	**J_n**	**J_n x L**	**J_r**	**J_r x L**	**J_a**	**J_a x L**
Biotite-Quartz-Feldspar Gneiss	A7/R1	4.5	20	90	4	18	0.75	3.375	10	45
Pegmatite	A7/R1	17.0	20	340	4	68	0.75	12.75	10	170
TOTAL:		21.5		430		86		16.125		215
			RQD_w	**20.0**	**$J_{n(w)}$**	**4.0**	**$J_{r(w)}$**	**0.8**	**$J_{a(w)}$**	**10.0**

Overcut (292L) - Parameter Based Q' Approach										
Rock Type	**A/R value**	**Length (m)**	**RQD**	**RQD x L**	**J_n**	**J_n x L**	**J_r**	**J_r x L**	**J_a**	**J_a x L**
Biotite-Quartz-Feldspar Gneiss	A7/R1	4.5	75	337.5	9	40.5	0.75	3.375	4	18
Pegmatite	A7/R1	17.0	25	425	4	68	0.75	12.75	10	170
TOTAL:		21.5		762.5		108.5		16.125		188
			RQD_w	**35.5**	**$J_{n(w)}$**	**5.0**	**$J_{r(w)}$**	**0.8**	**$J_{a(w)}$**	**8.7**

Overcut (292L) - RMR_{76} to Q' Equation Based Conversion Approach						
Rock Type	**A/R value**	**Length (m)**	**RQD**	**RQD x L**	**Q'**	**Q' x L**
Biotite-Quartz-Feldspar Gneiss	A7/R1	4.5	69	310.5	3.6	16.2
Pegmatite	A7/R1	17.0	38	646	0.3	5.1
TOTAL:		21.5		956.5		21.3
			RQD_w	**44.5**	**$Q'_{(w)}$**	**1.0**

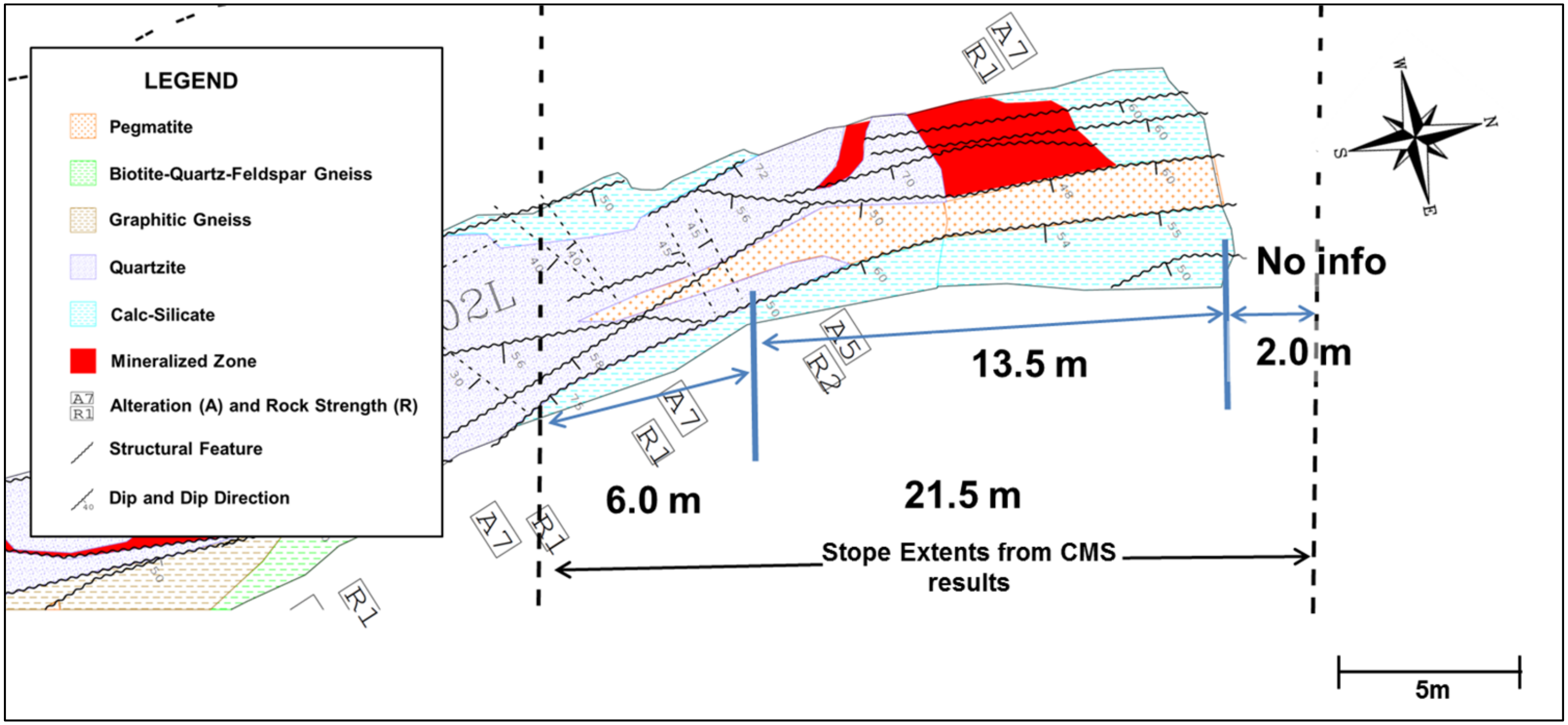


Figure 7-17 Geological Mapping and A/R extents for 302L (302-2 stope undercut).

Table 7-3 Calculations of the weighted RQD, J_n, J_r, and J_a values for the Sutton Q' Approach and Parameter Based Q' Approach, and the weighted RQD and Q' values for the RMR_{76} to Q' Equation Based Conversion Approach for the 302-2 stope undercut (302L)

Undercut (302L) - Sutton Q' Approach										
Rock Type	A/R value	Length (m)	RQD	RQD x L	J_n	J_n x L	J_r	J_r x L	J_a	J_a x L
Calc-Silicate	A7/R1	6.0	20	120	4	24	0.75	4.5	10	60
Calc-Silicate	A5/R2	13.5	75	1012.5	7.5	101.25	1.5	20.25	6	81
TOTAL:		19.5		1132.5		125.25		24.75		141
			RQD_w	58.1	$J_{n(w)}$	6.4	$J_{r(w)}$	1.3	$J_{a(w)}$	7.2

Undercut (302L) - Parameter Based Q' Approach										
Rock Type	A/R value	Length (m)	RQD	RQD x L	J_n	J_n x L	J_r	J_r x L	J_a	J_a x L
Calc-Silicate	A7/R1	6.0	25	150	4	24	0.75	4.5	10	60
Calc-Silicate	A5/R2	13.5	60	810	7.5	101.25	1.5	20.25	4	54
TOTAL:		19.5		960		125.25		24.75		114
			RQD_w	49.2	$J_{n(w)}$	6.4	$J_{r(w)}$	1.3	$J_{a(w)}$	5.8

Undercut (302L) - RMR_{76} to Q' Equation Based Conversion Approach						
Rock Type	A/R value	Length (m)	RQD	RQD x L	Q'	Q' x L
Calc-Silicate	A7/R1	6.0	38	228	0.3	1.8
Calc-Silicate	A5/R2	13.5	63	850.5	2.5	33.75
TOTAL:		19.5		1078.5		35.55
			RQD_w	55.3	$Q'_{(w)}$	1.8

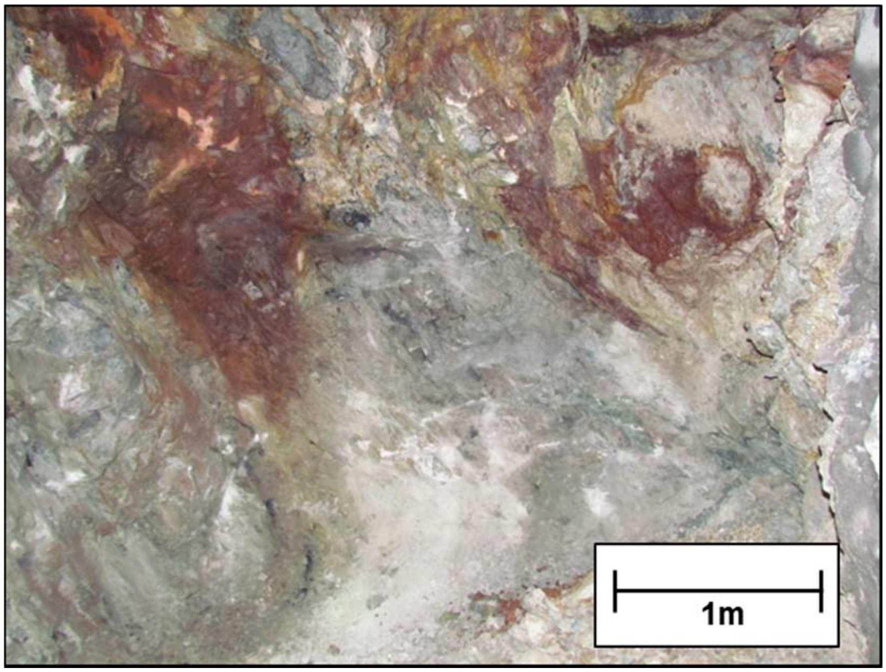


Figure 7-18 Photograph of the drift face on 292L (302-2 Overcut) at Ring 9 (looking north).

Figure 7-19 Photograph of full 302L (302-2 Undercut) round taken at Ring 9 (looking north).

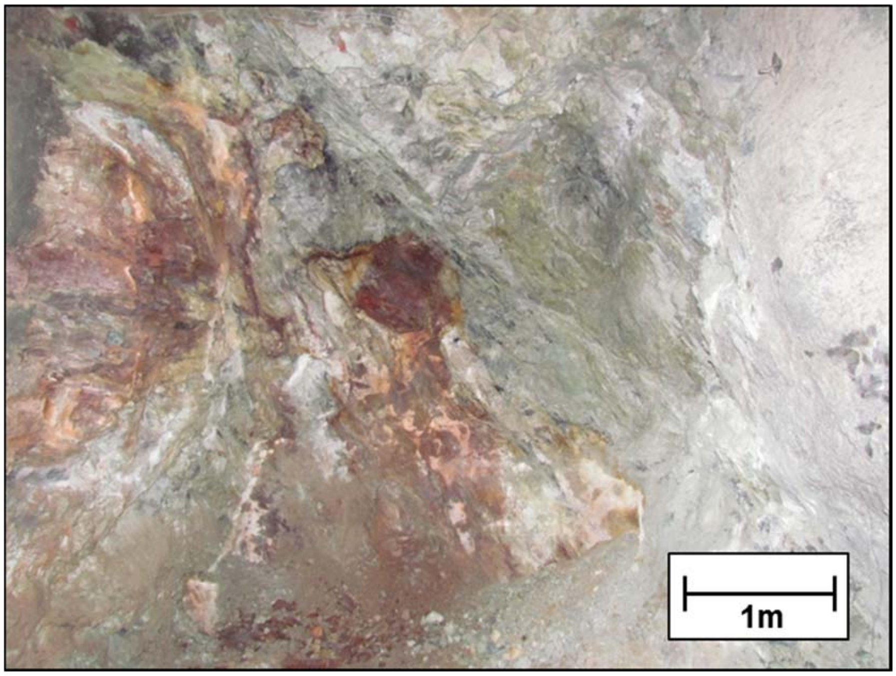


Figure 7-20 Photograph of the face on 302L (302-2 Undercut) at Ring 9 (looking north).

Three diamond drill holes intersected the stope block. The hole numbers and corresponding immediate hanging wall RQD values are as follows:

DDH-2167 = 77% (over 3 m)

DDH-2245 = 87% (over 3 m)

DDH-2251 = 30% (over 3 m)

The $RQD_{average}$ is the arithmetic mean of the overcut RQD, the undercut RQD, and the three RQD values from the diamond drill holes for all three approaches. The weighted input parameters for the three approaches are found in Table 7-4.

Table 7-4 Summary of Q' values from Sutton Q', Parameter Based Q', and RMR_{76} to Q' Equation Based Conversion approaches.

	Sutton Q' Approach	Parameter Based Q' Approach	RMR_{76} to Q' Equation Based Conversion Approach
$RQD_{average}$	54.4	55.7	58.8
$(RQD_{o/c} + RQD_{u/c})/2$			49.9
$J_{n(w)}$	5.2	5.7	
$J_{r(w)}$	1.0	1.0	
$J_{a(w)}$	8.6	7.3	
$Q'_{(w)}$			1.4
Q'	**1.2**	**1.3**	**1.7**

The N' factors were as follows for all three approaches:

A = 1.0 (hanging wall in relaxation)

B = 0.2 (angle between the hanging wall dip and the primary joint set is 10°)

C = 3.5 (average hanging wall dip of 41.8° from 45°, 44°, 45°, 40°, 40°, and 37° as measured from the hanging wall drill holes in the primary rings)

The hydraulic radius was calculated to be 6.0 m based upon a measured strike length of 21.5 m, and an average up-dip hanging wall length of 26.8 m.

An N' value of 0.86 was calculated from the Sutton A/R to Q' approach, 0.95 for the Parameter Based Q' approach, and 1.17 for the RMR_{76} to Q' approach. The hydraulic radius is the same for all three scenarios.

7.3.4 **Summary of Results**

Based on plotting these values on the Modified Stability Graph (Figure 7-21) an ELOS dilution of 2.0 m was predicted from both the Sutton Q' and Parameter Based Q' approaches. The RMR_{76} to Q' approach predicted an ELOS dilution of 1.8 m. The average measured hanging wall ELOS was 3.2 m. Based on Figure 7-21, the predicted ELOS was between 1.9 and 2.2 metres.

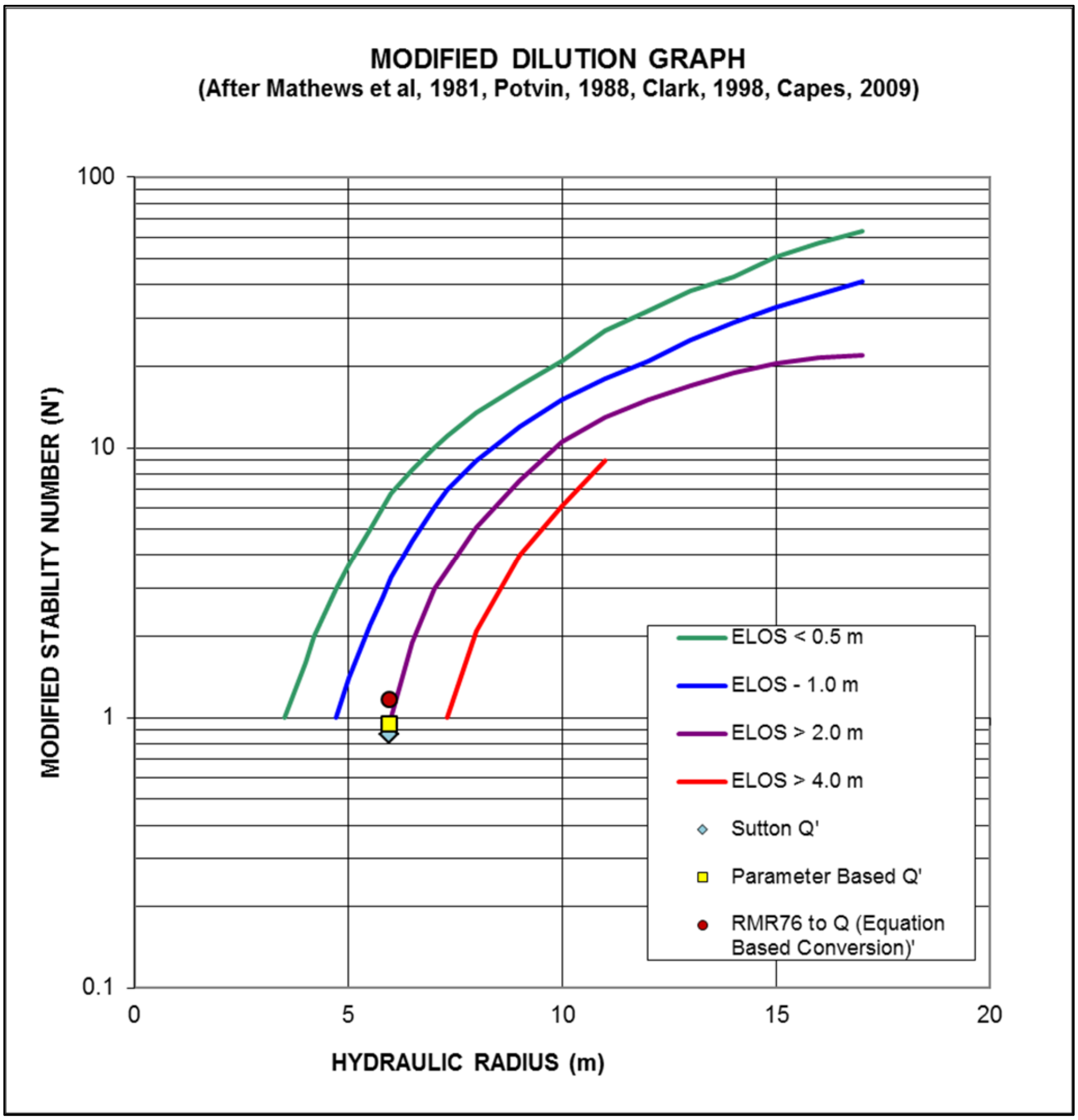


Figure 7-21 Modified Dilution Graph illustrating the predicted dilutions from the A/R observations for the Sutton Q', Parameter Based Q', and RMR_{76} to Q' Equation Based approaches.

Chapter 8 – Case Histories

In total, 27 stopes were analyzed using the method described in Section 7.3. A long section view of the Eagle Point Mine showing the lateral development, vertical development, and stopes, including the stopes specifically analyzed for this project, is presented in Figure 8-1. The Sutton Q' values, Parameter Based Q' values, and RMR_{76} to Q' Equation Based Conversion values were used to predict the stope dilution for each stope, and were compared to the measured dilution values from the cavity monitoring survey.

8.1 Comparison of Dilution Predictions for Sutton Q', Parameter Based Q', and RMR_{76} to Q' Equation Based Conversion

Table 8-1 presents the summary of the Q' and Modified Stability Number, N' values from the Sutton Q', Parameter Based Q', and RMR_{76} to Q' Equation Based Conversion approaches for the 27 case histories. Table 8-2 presents the summary of the Hydraulic Radii, Dilution predictions, and measured dilution values from the Sutton Q', Parameter Based Q', and RMR_{76} to Q' Equation Based Conversion approaches for all of the case histories. Ranges for the measured ELOS dilution values corresponding to the ELOS lines on the Modified Dilution Graph were included for reference. The case history stope reconciliations are found in Appendix E.

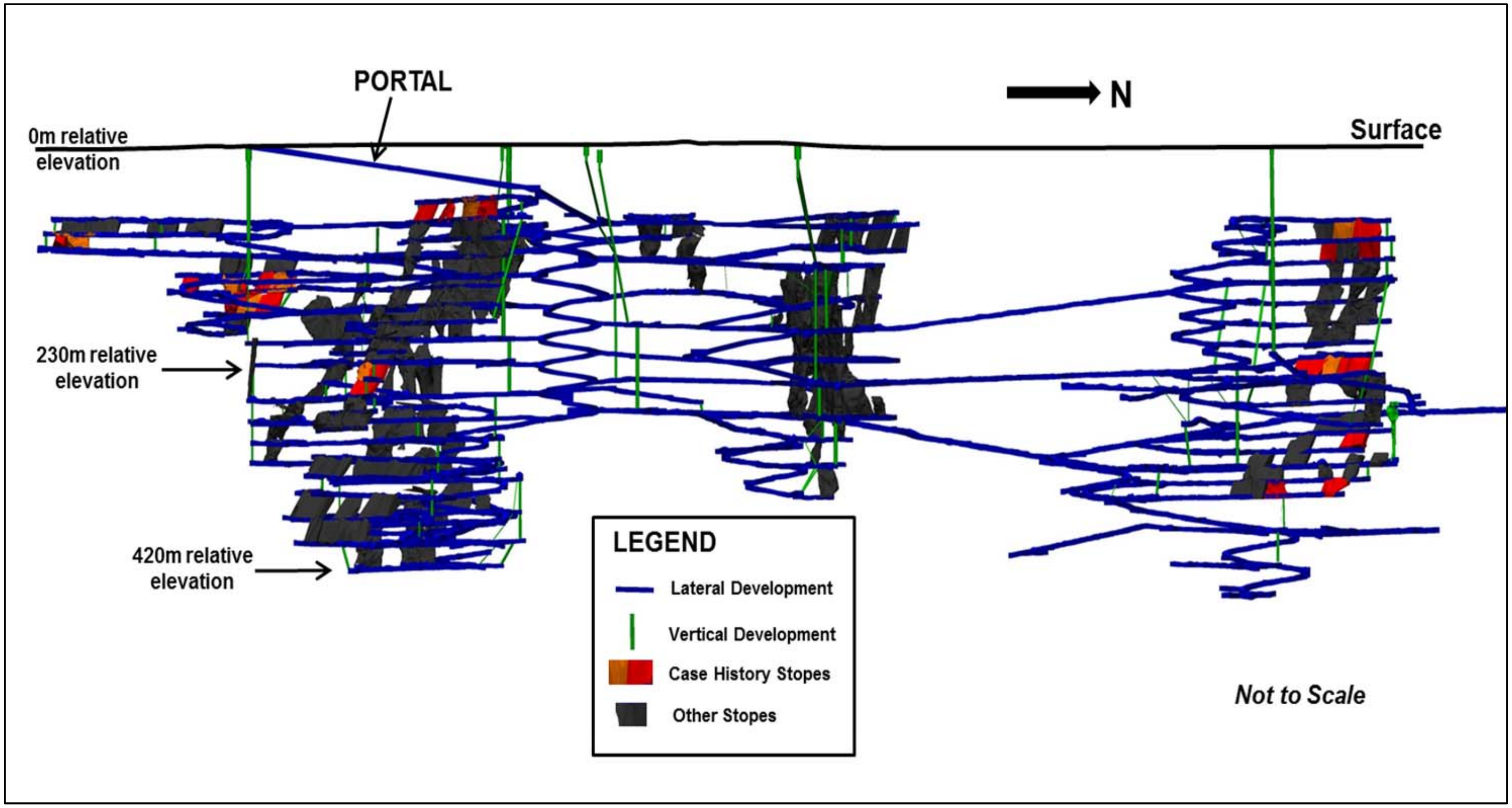


Figure 8-1 Isometric mine long section looking west. Lateral development, vertical development, mined out stopes, and the case history stopes are shown.

Table 8-1 Stope case history summary table for the Q' and Modified Stability Number, N' values from the Sutton Q', Parameter Based Q', and RMR_{76} to Q' Equation Based approaches.

Stope Name	Q'			Modified Stability Number, N'		
	Sutton Q' Approach	Parameter Based Q' Approach	RMR_{76} to Q' Approach	Sutton Q' Approach	Parameter Based Q' Approach	RMR_{76} to Q' Approach
100-045	6.5	8.2	8.7	6.0	7.6	8.1
362-4	1.3	1.2	2.4	1.2	1.2	2.2
100-075	2.1	3.2	7.6	3.3	5.0	11.8
100-085	2.2	3.6	7.8	2.8	4.5	9.9
245-065	1.7	2.9	6.5	1.4	2.4	5.2
102-1	1.6	2.7	5.4	1.7	2.9	5.7
122-3	2.6	4.3	9.0	3.8	6.3	13.1
102-2	1.7	2.6	5.8	2.1	3.2	7.1
150-880	1.6	1.6	3.2	1.6	1.6	3.1
150-900	2.3	2.4	3.6	2.0	2.0	3.1
100-015	2.6	4.0	8.7	2.3	3.5	7.6
170-900	2.2	2.5	4.3	1.9	2.1	3.6
252-1	2.1	3.0	6.6	1.9	2.7	6.0
122-1	1.3	2.3	4.5	2.0	3.5	6.9
260-065	1.6	2.7	6.2	1.2	2.0	4.6
252-4	0.9	1.2	3.4	0.8	1.1	3.0
245-055	1.1	1.9	4.2	0.7	1.2	2.6
150-835	1.2	2.1	4.7	1.1	1.9	4.2
125-675	2.6	2.9	4.4	1.7	1.9	2.9
150-860	1.6	1.7	2.6	1.7	1.8	2.8
252-3	1.8	3.2	7.1	1.6	2.7	6.0
150-870	1.6	1.9	1.9	1.5	1.7	1.7
125-695	2.6	2.9	7.4	1.7	1.9	4.9
170-870	2.2	2.5	3.5	1.8	2.1	2.8
302-2	1.2	1.3	1.7	0.9	0.9	1.2
362-3	0.8	0.8	0.9	0.4	0.4	0.5
170-860	1.6	1.8	2.3	1.3	1.5	1.9

Table 8-2 Stope case history summary table for the Hydraulic Radii, Dilution predictions, and measured dilution values from the Sutton Q', Parameter Based Q', and RMR_{76} to Q' Equation Based approaches. Measured Dilution Ranges are also included for reference.

Stope Name	Hydraulic Radius (m)	Dilution Prediction (m)			Measured ELOS (m)	Measured Dilution Range (m)
		Sutton Q' Approach	Parameter Based Q' Approach	RMR_{76} to Q' Approach		
100-045	5.9	0.6	0.4	0.4	0.3	0 to 0.5 m
362-4	4.6	0.9	0.9	0.6	0.4	
100-075	5.7	0.9	0.6	0.1	0.1	
100-085	5.6	1.0	0.6	0.1	0.8	0.5 to 1 m
245-065	5.5	1.3	1.0	0.5	0.8	
102-1	6.4	2.0	1.5	0.8	0.7	
122-3	5.4	0.7	0.4	0.0	1.0	
102-2	6.2	1.7	1.2	0.6	0.7	
150-880	5	0.9	0.9	0.6	1.9	1 to 2 m
150-900	6	1.5	1.5	1.1	1.6	
100-015	6.9	2.2	1.7	0.8	1.5	
170-900	6.5	2.0	1.9	1.3	1.1	
252-1	7.4	3.4	2.7	1.3	1.6	
122-1	6	1.5	0.9	0.5	1.7	
260-065	6.7	2.9	2.1	1.1	1.8	
252-4	6.4	2.7	2.5	1.5	1.8	
245-055	5.2	1.5	1.2	0.8	2.7	2 to 3 m
150-835	5.3	1.3	1.0	0.5	2.6	
125-675	5.6	1.3	1.2	0.9	2.7	
150-860	5	0.9	0.9	0.7	2.8	
252-3	6	1.7	1.2	0.6	2.9	
150-870	5.4	1.2	1.1	1.1	3.0	3 to 4 m
125-695	6.3	1.9	1.8	0.9	3.0	
170-870	5.2	1.0	0.9	0.8	3.1	
302-2	6.0	2.0	2.0	1.8	3.2	
362-3	6.2	2.8	2.8	2.8	4.3	> 4 m
170-860	7.6	4.0	4.0	3.6	5.4	

The percentage of each hanging wall rock present from the geology drift mapping data for each case study is presented in Table 8-3. The difference between each of the predicted dilution values (Sutton Q', Parameter Based Q', and RMR_{76} to Q' Equation Based Conversion) and the measured dilution for each stope was calculated. These differences were plotted in Figure 8-2 by the percentage of gneissic hanging wall rock to evaluate the effect of the predominant hanging wall rock type on the prediction. A negative difference indicates that there was more measured dilution than predicted dilution, and a positive difference indicates that the predicted dilution was greater than the measured dilution. The dilution predictions for most of the stopes having a low gneissic, or high pegmatoidal, hanging wall rock content were underestimated compared to the measured dilution values. The dilution predictions for most of the stopes having a high gneissic, or low pegmatoidal, hanging wall rock content had values that over- and underestimated the dilution, so no trends were apparent for this comparison.

Table 8-3 Percentage of hanging wall rock type for case study stopes.

Stope Name	Hydraulic Radius (m)	Percentage of Hanging Wall Rock	
		Gneissic	Pegmatoidal
100-045	5.9	59%	41%
362-4	4.6	2%	98%
100-075	5.7	81%	19%
100-085	5.6	98%	2%
245-065	5.5	98%	2%
102-1	6.4	73%	27%
122-3	5.4	100%	0%
102-2	6.2	92%	8%
150-880	5	15%	85%
150-900	6	6%	94%
100-015	6.9	79%	21%
170-900	6.5	35%	65%
252-1	7.4	90%	10%
122-1	6	81%	19%
260-065	6.7	94%	6%
252-4	6.4	75%	25%
245-055	5.2	98%	2%
150-835	5.3	100%	0%
125-675	5.6	22%	78%
150-860	5	7%	93%
252-3	6	92%	8%
150-870	5.4	0%	100%
125-695	6.3	16%	84%
170-870	5.2	9%	91%
302-2	6.0	11%	89%
362-3	6.2	3%	97%
170-860	7.6	4%	96%

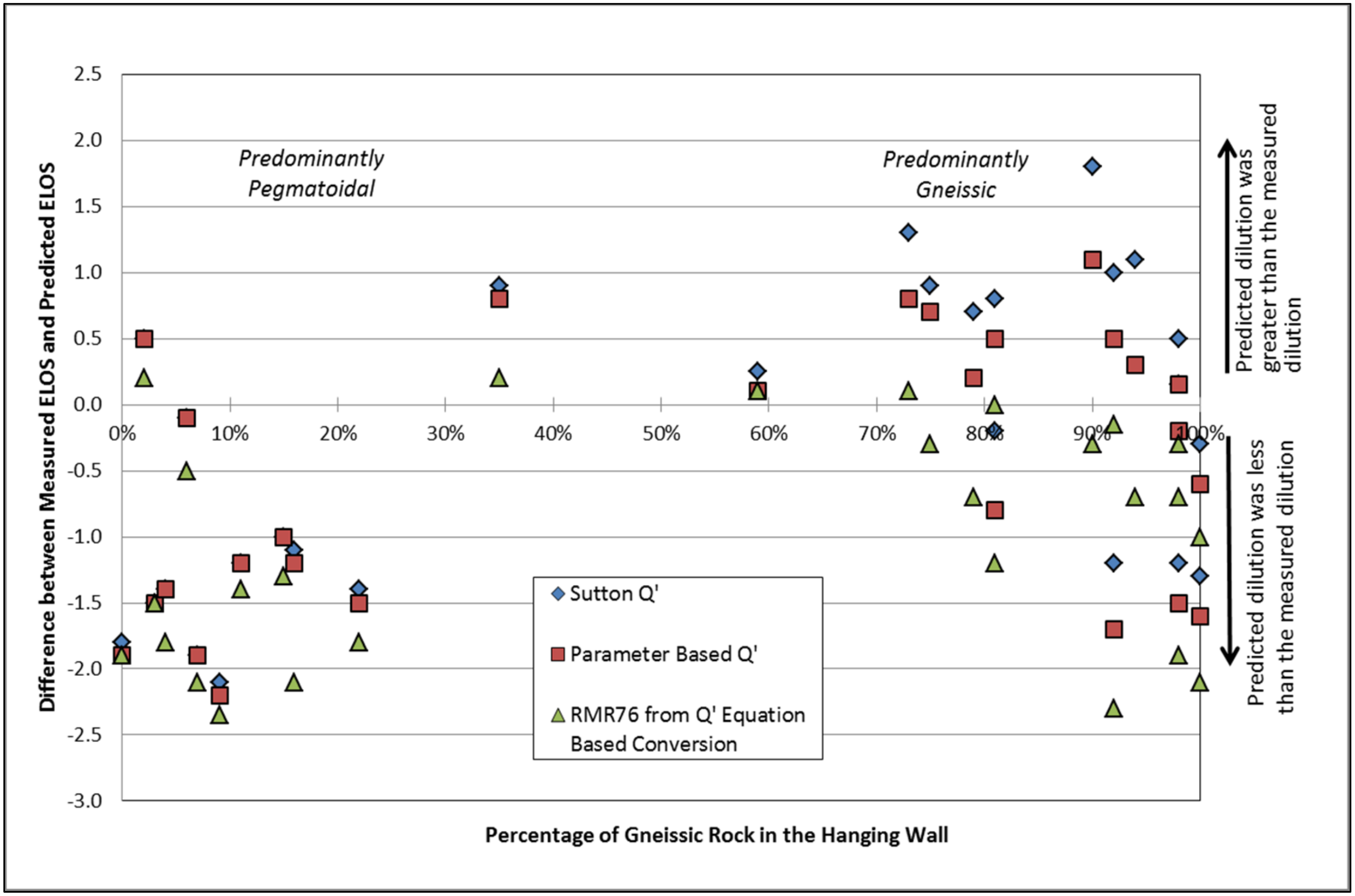


Figure 8-2 Difference between the measured and predicted ELOS values by percentage of gneissic hanging wall rock.

To compare the three prediction approaches relative to each other, a cumulative sum graph was generated as shown in Figure 8-3. The cumulative sums technique is used to look for trends in sequentially ordered data (Piteau and Russell, 1971). The difference between a measured value and the expected or average value for a collection of data is determined for each data point. The sum of these differences is then plotted by percentage of gneissic hanging wall rock from 0% to 100%. The absolute difference between the predicted dilution and the measured dilution was calculated for each Q' approach. Each absolute difference was added to the previous sum to create the cumulative sum graph.

As many of the values were concentrated near the 0% and 100% gneissic hanging wall rock content ends of the graph, the cumulative sum for each stope was also plotted sequentially, as well as based on the percentage of gneissic rock content. The cumulative sum values were plotted a second time based on increasing gneissic content in Figure 8-4. The line approximately corresponding to 50% gneissic hanging wall rock content is shown for reference.

What may be interpreted from these two graphs is that the Parameter Based Q' approach gives a slightly lower total absolute dilution prediction error for the 27 case histories, compared to the Sutton Q' and RMR_{76} to Q' Equation Based Conversion approaches. The RMR_{76} to Q' Equation Based Conversion approach gave the largest absolute cumulative error.

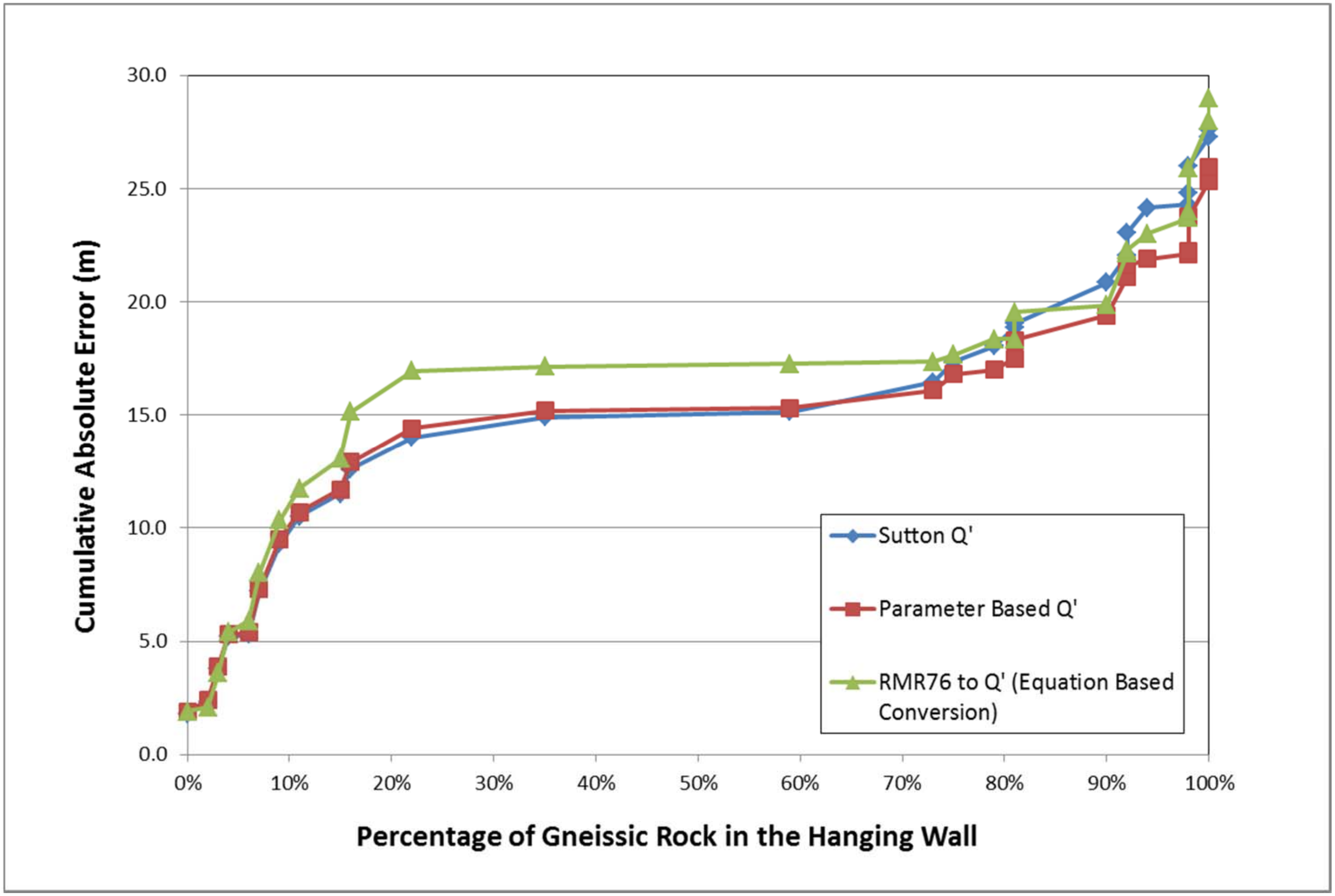


Figure 8-3 Cumulative absolute error between the measured and predicted dilution values for the three Q' approaches. Stopes are arranged from 0% gneissic hanging wall rock to 100% gneissic hanging wall rock.

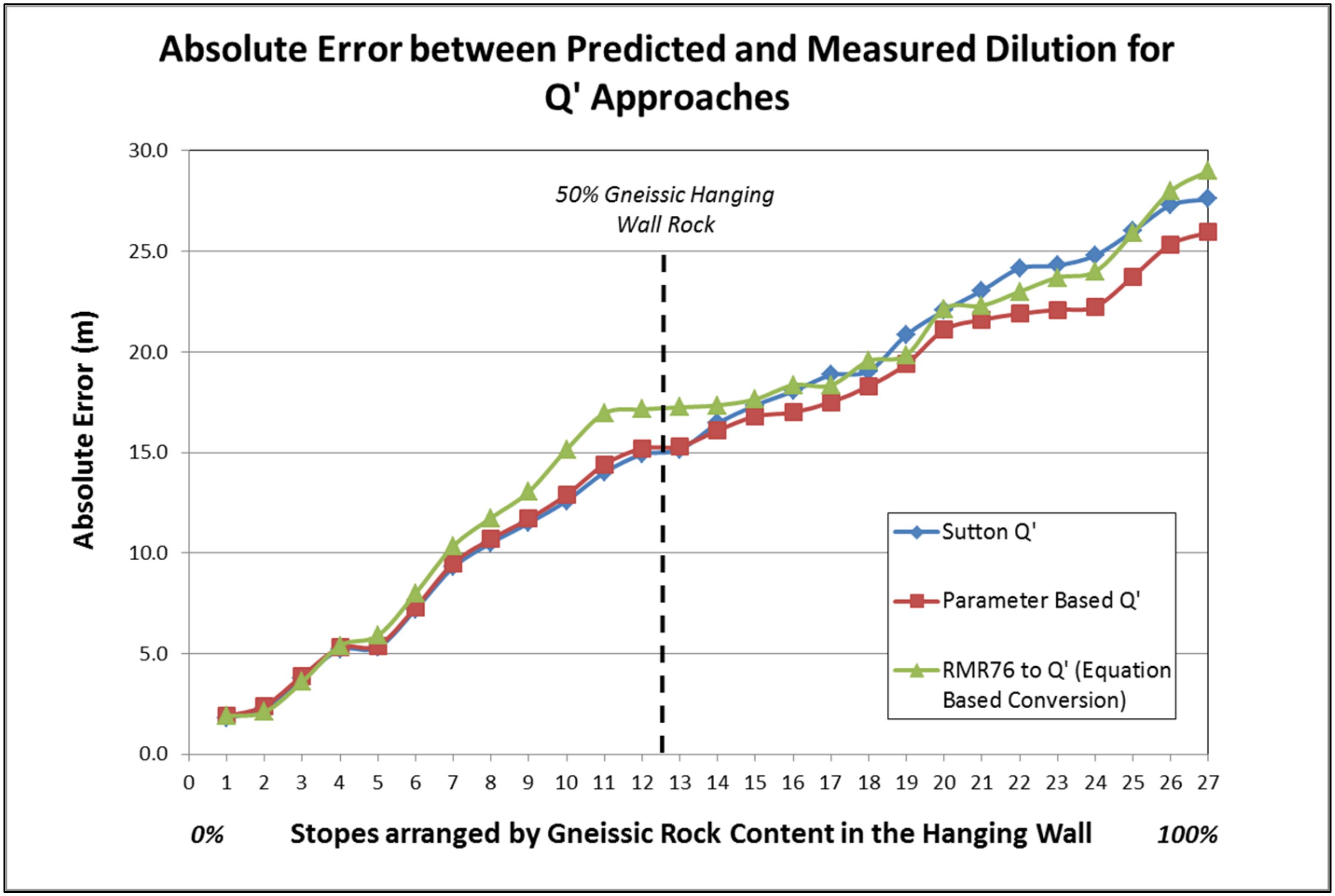


Figure 8-4 Cumulative absolute error between the measured and predicted dilution values for the three Q' approaches. Stopes are arranged in relative order from 0% gneissic hanging wall rock to 100% gneissic hanging wall rock.

Two additional cumulative sum graphs were analyzed with the cumulative sum of the difference between the predicted and measured dilutions. A negative value indicates that the predicted dilution was less than the measured dilution. Figure 8-5 shows the cumulative sum of the differences between the predicted and measured dilution values for the stope case histories arranged by the percentage of gneissic hanging wall rock. As with Figure 8-3, this graph also has many of the values concentrated near the 0% and 100% gneissic hanging wall rock content ends of the graph. Figure 8-6 shows the stope case histories evenly spaced along the horizontal axis, arranged from 0% to 100% gneissic hanging wall rock content.

A hanging wall rock content of less than 50% gneiss indicates that the hanging wall rock is predominantly pegmatoidal. A downward trend in the cumulative sum line indicates that the predicted dilution values are consistently lower than the measured dilution vales, which could be due to an overestimation of the quality of the rock mass. An upward trend in the cumulative sum line indicates that the predicted dilution values are consistently higher than the measured dilution values, which could be due to an underestimation of the quality of the rock mass. A horizontal line on the graph would indicate that there was little difference between the predicted and measured dilution values, and could indicate that the rock mass has been characterized appropriately.

Figure 8-6 shows an interesting trend for the stope hanging walls that were predominantly pegmatoidal. Between the first and eleventh stope (circled in Figure 8-6), there is a linear trend with the cumulative dilution prediction. The cumulative dilution errors for the Sutton Q', Parameter Based Q', and RMR_{76} to Q' Equation Based Conversion methods are 13 m, 13 m, and 17 m respectively. This corresponds to an average dilution underestimation of 1.2 m, 1.2 m and 1.5 m for the first eleven predominantly pegmatoidal hanging wall case histories.
This could also indicate that the rock classification Q' estimates for these pegmatoidal rocks are generally overestimated for all three approaches.

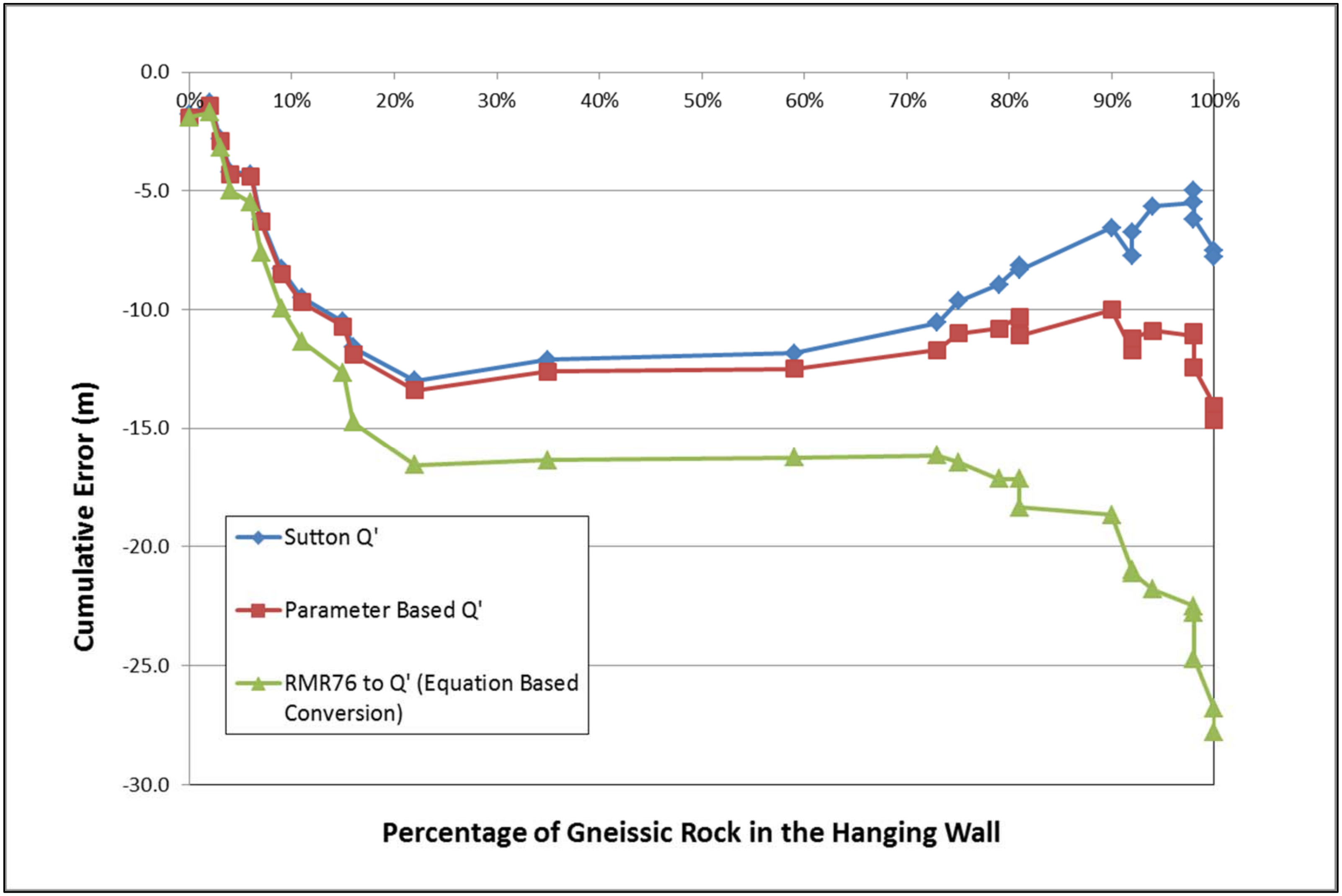


Figure 8-5 Cumulative error between the measured and predicted dilution values for the three Q' approaches. Stopes are arranged from 0% gneissic hanging wall rock to 100% gneissic hanging wall rock.

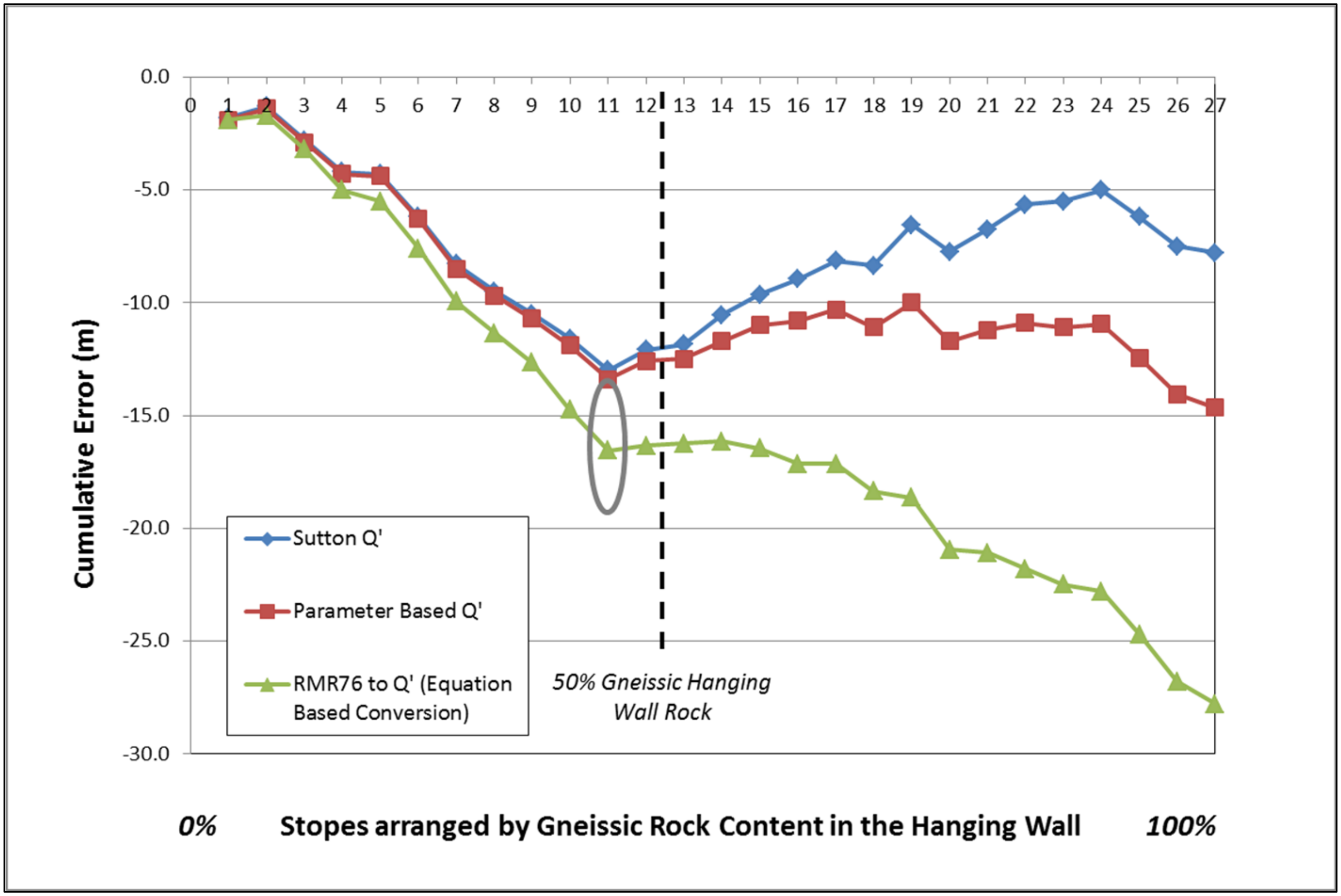


Figure 8-6 Cumulative error between the measured and predicted dilution values for the three Q' approaches. Stopes are arranged in relative order from 0% gneissic hanging wall rock to 100% gneissic hanging wall rock.

At approximately 50% gneissic hanging wall rock content, the Sutton Q' values begin to trend upwards, indicating that the predicted dilution is generally greater than the measured dilution for the gneissic hanging wall rock stopes. This could indicate that the gneissic rock qualities are underestimated using the Sutton Q' approach. At the same 50% gneissic hanging wall rock content, the RMR_{76} to Q' Equation Based Conversion values continue to trend downwards, indicating that the predicted dilution is generally less than the measured dilution for the gneissic hanging wall rock stopes. This could indicate that the gneissic rock qualities are overestimated using the RMR_{76} to Q' Equation Based Conversion approach. The Parameter Based Q' cumulative sum line tends to trend horizontally for the rock types that are greater than 50% gneissic hanging wall rock content. This may indicate that the gneissic rock qualities are appropriate for the Parameter Based Q' approach.

Although general inferences can be made about the quality of the rock type, the previous graphs include both poor quality rock masses (Q' ≤ 2.5) and better quality rock masses (Q' > 2.5). The data was separated by a Parameter Based Q' value of 2.5 and the results for Q' ≤ 2.5 are presented in Figure 8-7, and the results for Q' > 2.5 are presented in Figure 8-8. The cumulative sum differences in dilution prediction were plotted by the stope case histories, which were evenly spaced along the horizontal axis from 0% to 100% gneissic hanging wall rock content.

Figure 8-7 shows that almost all the case histories of predominantly pegmatoidal rocks that were in poor quality rock (Q' ≤ 2.5) underestimated the actual amount of dilution. It appears that classification values in the pegmatoidal rocks were too high for all of the classification approaches tried. It is recognized that poor quality rocks are difficult to obtain realistic rock classification values. Figure 8-8 shows that almost all of the case histories of predominantly gneissic rocks were in better quality rock (Q' > 2.5), and the Parameter Based Q' dilution predictions were closest to the measured dilution values.

The Parameter Based Q' approach data points were used for further analysis.

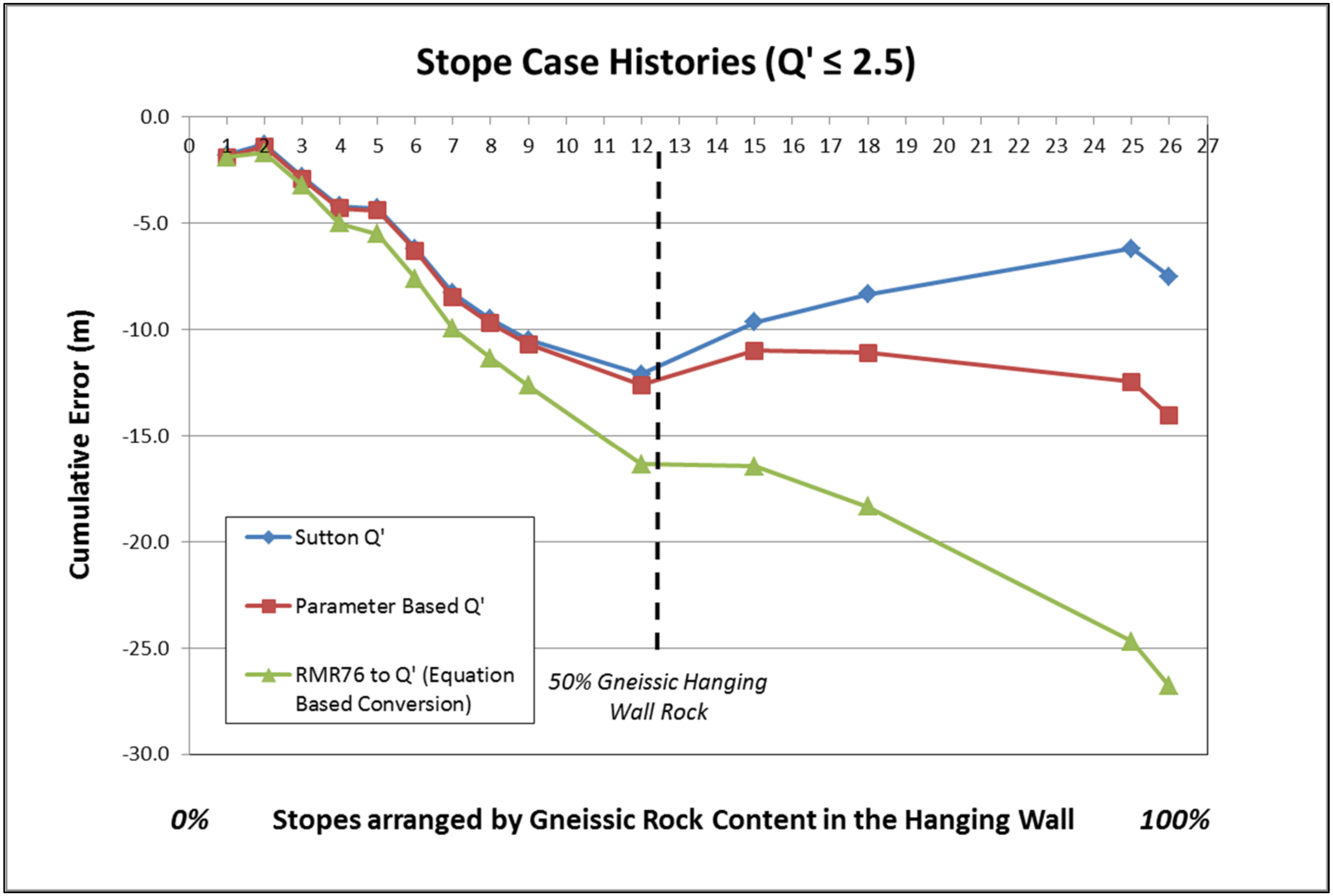


Figure 8-7 Cumulative error between the measured and predicted dilution values for the three Q' approaches and Q' ≤ 2.5. Stopes are arranged in relative order from 0% gneissic hanging wall rock to 100% gneissic hanging wall rock.

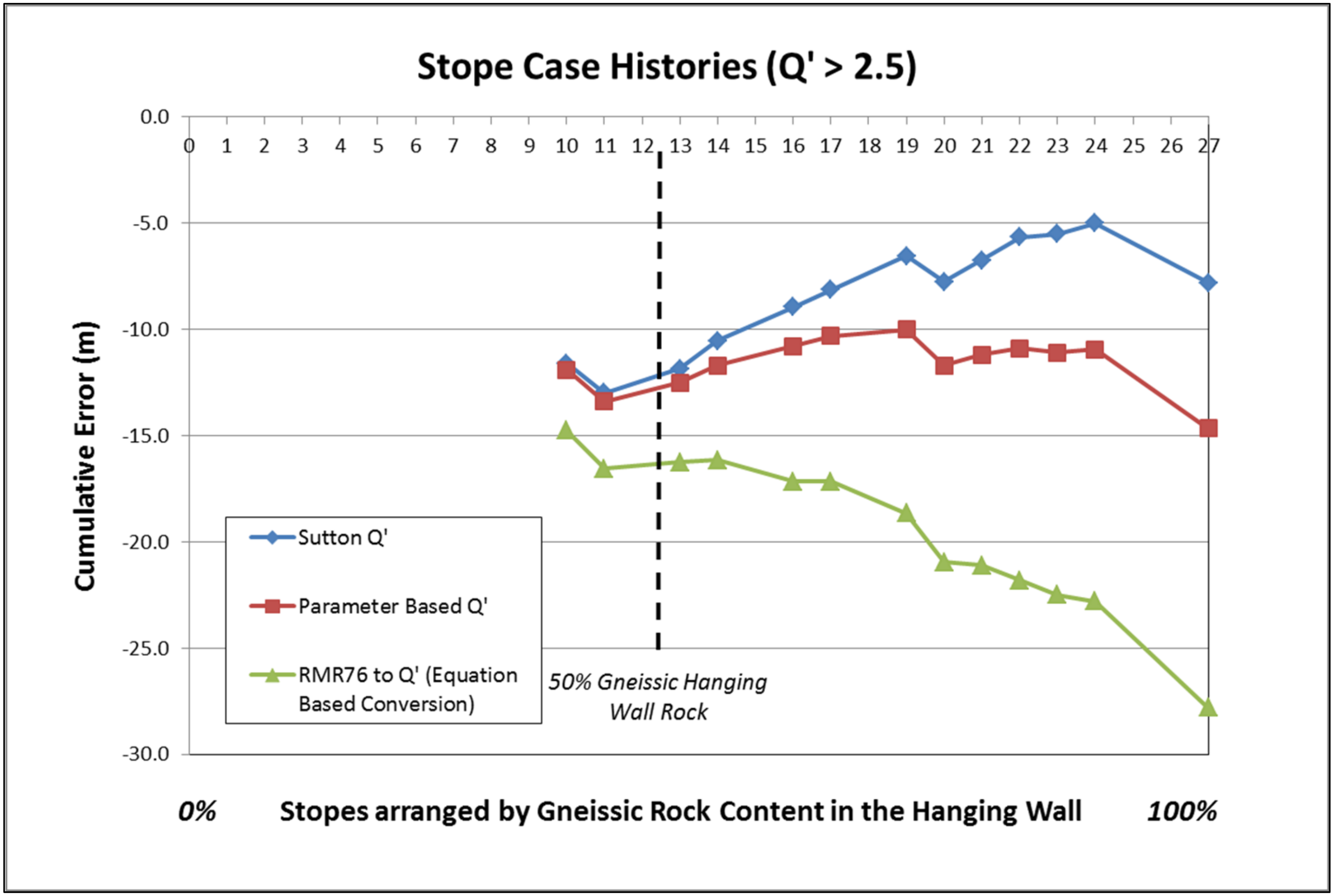


Figure 8-8 Cumulative error between the measured and predicted dilution values for the three Q' approaches and Q' > 2.5. Stopes are arranged in relative order from 0% gneissic hanging wall rock to 100% gneissic hanging wall rock.

8.2 Analysis of Parameter Based Q' Dilution Predictions

A summary of the stopes analyzed, the hydraulic radii, the Q' values from the Parameter Based Q' approach, the N' values, the predicted dilution, and the measured dilution are found in Table 8-4. The dilution results were grouped by measured dilution ranges in order to compare the predicted values and the measured values for the dilution, and the results are shown in Figure 8-9 to Figure 8-14. Several of the stopes were found to have been influenced by undercutting of the stope block, and these are outlined with grey circles. The dilution predictions were reasonable for the stopes with measured dilution between 0 and 2 m. The dilution predictions were too low for the stopes with measured dilution between 2 m and greater than 4 m. The dilution for these areas may have been influenced by some of the other factors which cause dilution, as outlined in Section 3.5, or the available rock mass data may not have been representative of the exposed hanging wall.

The Parameter Based Q' dilution results were analyzed in two groups, hanging walls consisting of more than 50% gneissic rock and hanging walls consisting of less than 50% gneissic rock. The assumption was made that stopes with less than 50% gneissic rock in the hanging wall are predominantly pegmatoidal in rock mass characteristics.

Table 8-4 Summary of the hydraulic radii, Parameter Based Q', Parameter Based N', predicted dilution, and measured dilution for the stopes analyzed. Measured dilution ranges are also included for reference.

Stope Name	Hydraulic Radius (m)	Parameter Based Q'	Parameter Based N'	Parameter Based Dilution Prediction (m)	Measured ELOS (m)	Measured Dilution Range (m)
100-045	5.9	8.2	7.6	0.4	0.3	0 to 0.5 m
362-4	4.6	1.2	1.2	0.9	0.4	
100-075	5.7	3.2	5.0	0.6	0.1	
100-085	5.6	3.6	4.5	0.6	0.8	0.5 to 1 m
245-065	5.5	2.9	2.4	0.95	0.8	
102-1	6.4	2.7	2.9	1.5	0.7	
122-3	5.4	4.3	6.3	0.4	1.0	
102-2	6.2	2.6	3.2	1.2	0.7	
150-880	5	1.6	1.6	0.9	1.9	1 to 2 m
150-900	6	2.4	2.0	1.5	1.6	
100-015	6.9	4.0	3.5	1.7	1.5	
170-900	6.5	2.5	2.1	1.9	1.1	
252-1	7.4	3.0	2.7	2.7	1.6	
122-1	6	2.3	3.5	0.9	1.7	
260-065	6.7	2.7	2.0	2.1	1.8	
252-4	6.4	1.2	1.1	2.5	1.8	
245-055	5.2	1.9	1.2	1.2	2.7	2 to 3 m
150-835	5.3	2.1	1.9	1	2.6	
125-675	5.6	2.9	1.9	1.2	2.7	
150-860	5	1.7	1.8	0.9	2.8	
252-3	6	3.2	2.7	1.2	2.9	
150-870	5.4	1.9	1.7	1.1	3.0	3 to 4 m
125-695	6.3	2.9	1.9	1.8	3.0	
170-870	5.2	2.5	2.1	0.9	3.1	
302-2	6.0	1.3	0.9	2	3.2	
362-3	6.2	0.8	0.4	2.8	4.3	> 4 m
170-860	7.6	1.8	1.5	4	5.4	

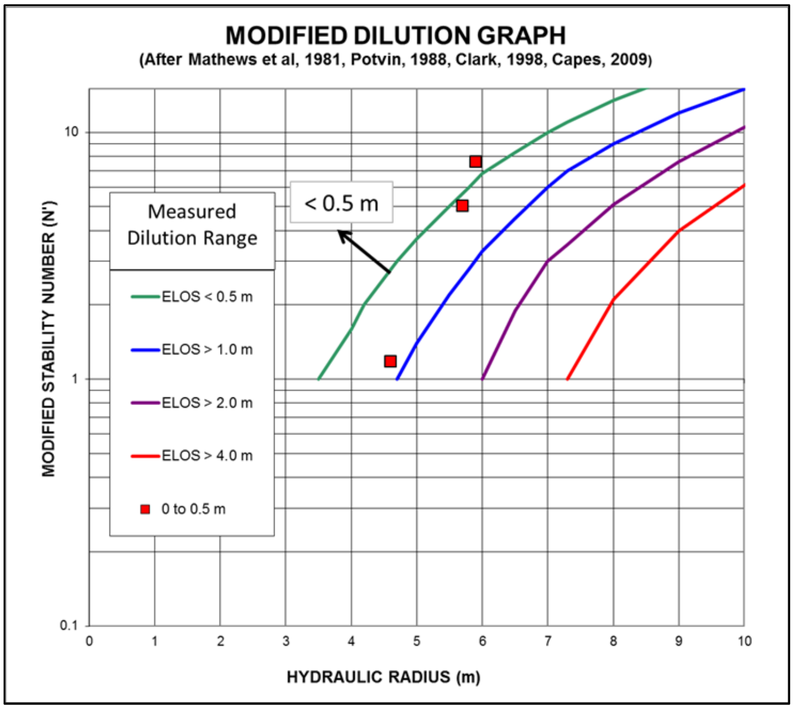


Figure 8-9 Modified Dilution Graph to illustrate the predicted dilutions for stopes which had actual dilutions of 0 to 0.5m.

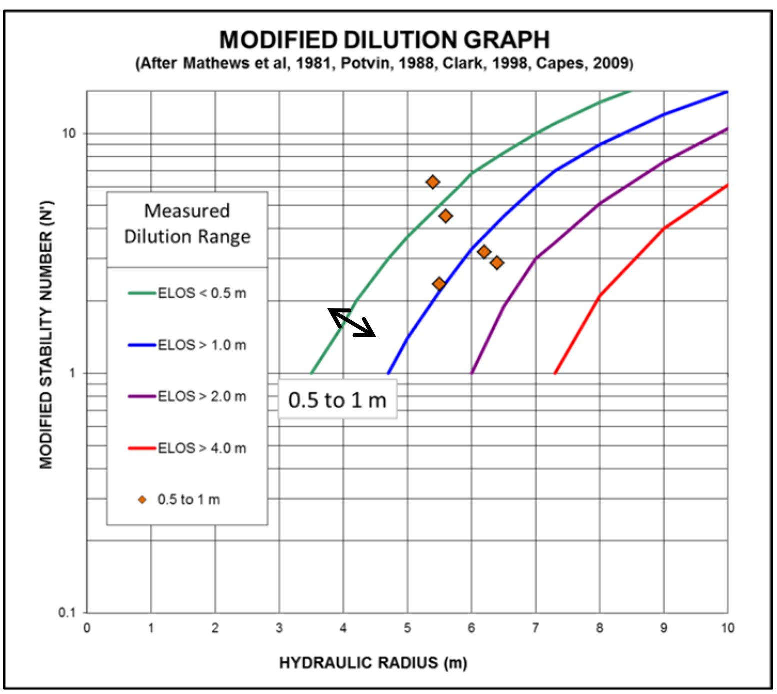


Figure 8-10 Modified Dilution Graph to illustrate the predicted dilutions for stopes which had actual dilutions of 0.5 to 1 m.

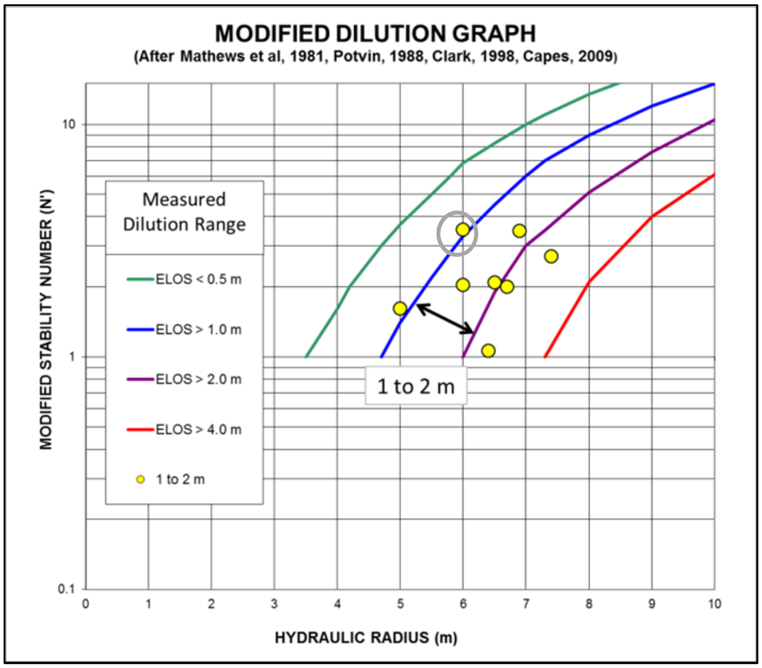


Figure 8-11 Modified Dilution Graph to illustrate the predicted dilutions for stopes which had actual dilutions of 1 to 2 m. Circled point indicates a stope influenced by undercutting.

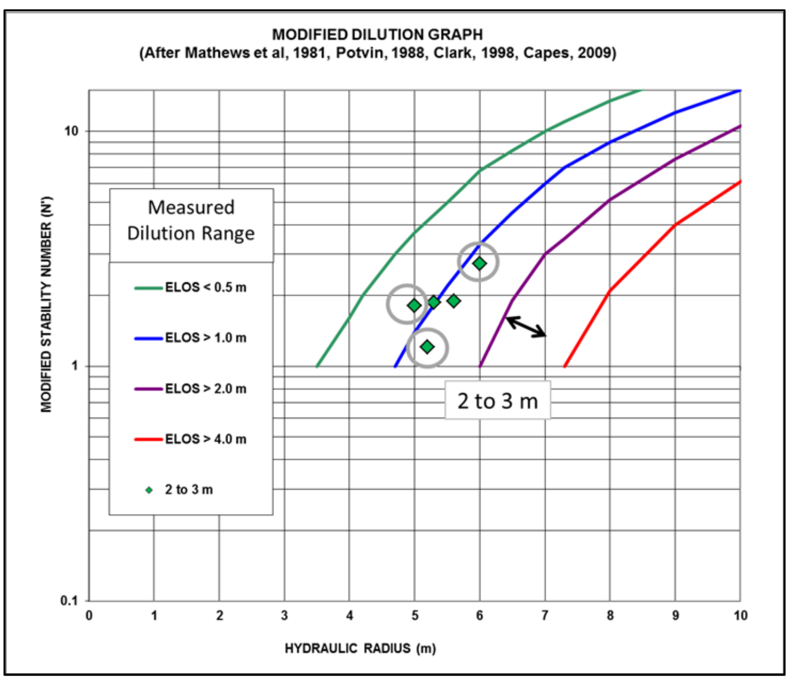


Figure 8-12 Modified Dilution Graph to illustrate the predicted dilutions for stopes which had actual dilutions of 2 to 3 m. Circled points indicate stopes influenced by undercutting.

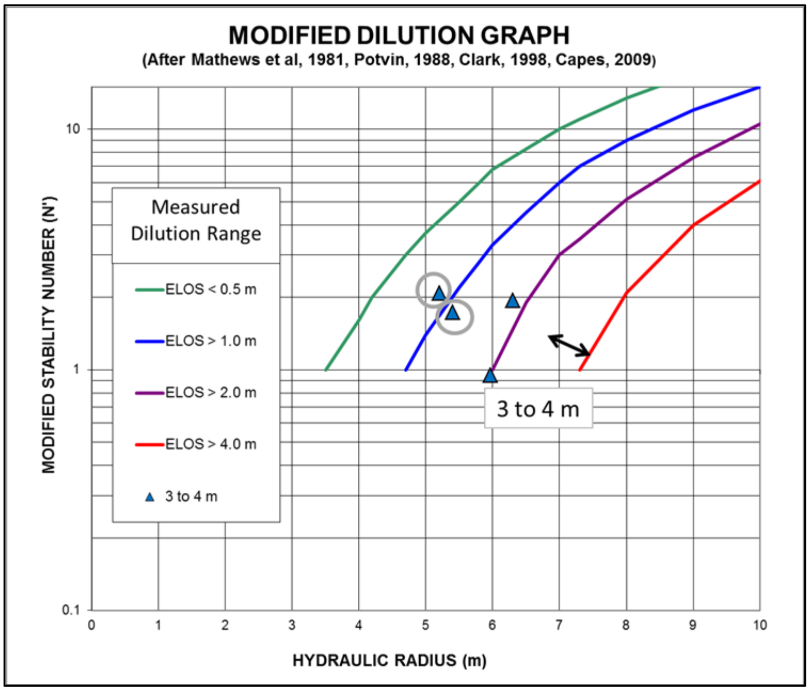


Figure 8-13 Modified Dilution Graph to illustrate the predicted dilutions for stopes which had actual dilutions of 3 to 4 m. Circled points indicate stopes influenced by undercutting.

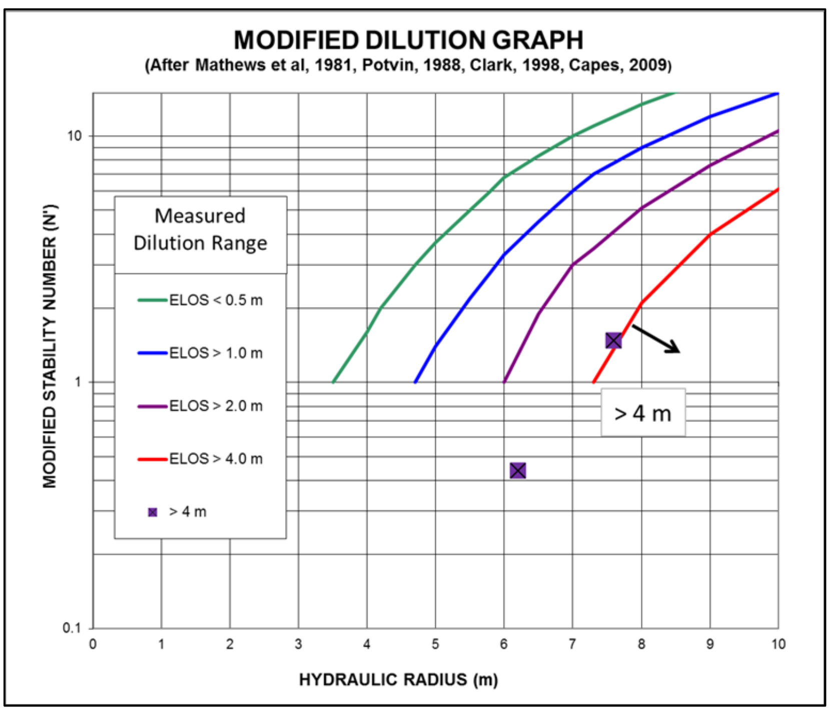


Figure 8-14 Modified Dilution Graph to illustrate the predicted dilutions for stopes which had actual dilutions of more than 4 m.

Figure 8-15 shows the difference between the predicted and measured dilution for each stope having less than 50% gneissic rock in the hanging wall. The difference in the dilution for each stope is plotted along the vertical axis, and by average Q' value along the horizontal axis. The equivalent pegmatoidal Q' values for several of the A/R categories are noted along the horizontal axis. The other A/R values are not shown as there were no case studies with average Q' values above 3.0.

Based on the difference between the predicted and measured stope dilutions, it appears that the Q' assumptions for the higher alteration and low strength pegmatoidal A/R categories are too high. As the N' value is proportional to Q', A, B, and C (Equation 3.1), and the A, B and C factors are constant for a given stope geometry. For any hydraulic radius on the Modified Stability Graph, the amount of predicted dilution decreases as the Stability Number increases. Therefore, had a lower N' value been used, based on a lower Q' value, those points in Figure 8-15 would have plotted closer to the 0 m of dilution prediction error.

Figure 8-16 shows the difference between the predicted and measured dilution for each stope for stopes with more than 50% gneissic rock in the hanging wall. The difference in the dilution for each stope is plotted along the vertical axis, and by average Q' value along the horizontal axis. The equivalent gneissic Q' values for several of the A/R categories are noted along the horizontal axis. The other A/R values are not shown as there were no case studies with average Q' values above 9.0. Based on the difference between the predicted and measured stope dilutions, no trends are apparent for the gneissic A/R categories. Unlike the pegmatoidal rock types, changes to the Q' assumptions would not have made a difference in obtaining dilution prediction errors closer to 0 m.

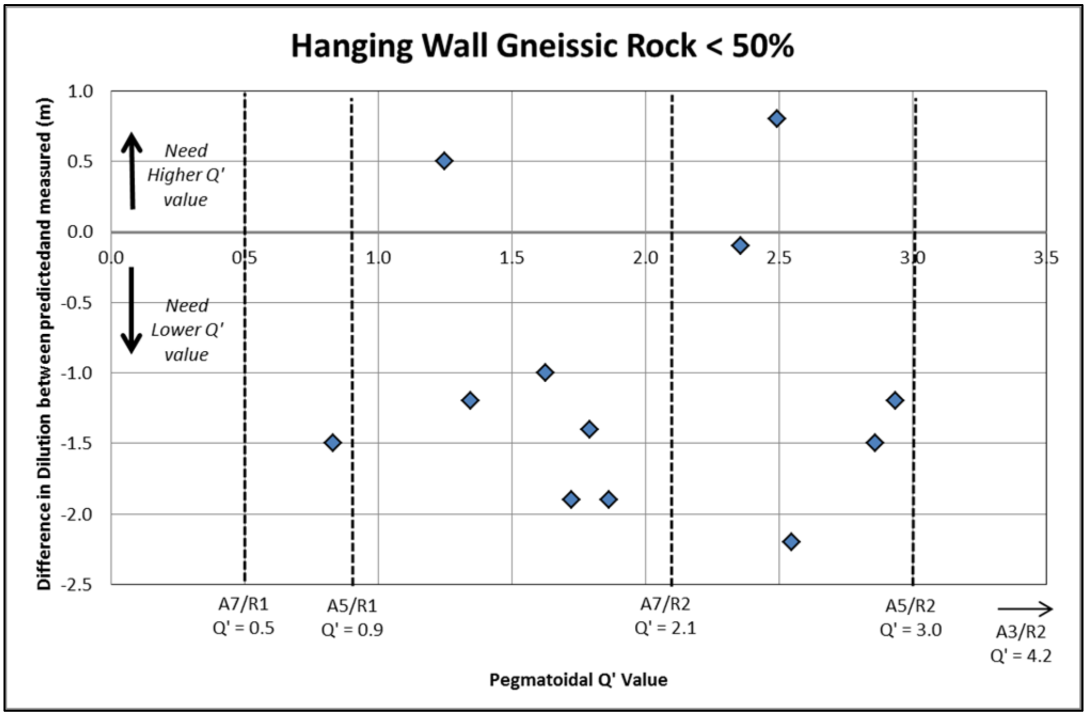


Figure 8-15 Difference between predicted and measured dilution versus Q' for stopes with less than 50% gneissic hanging wall rock.

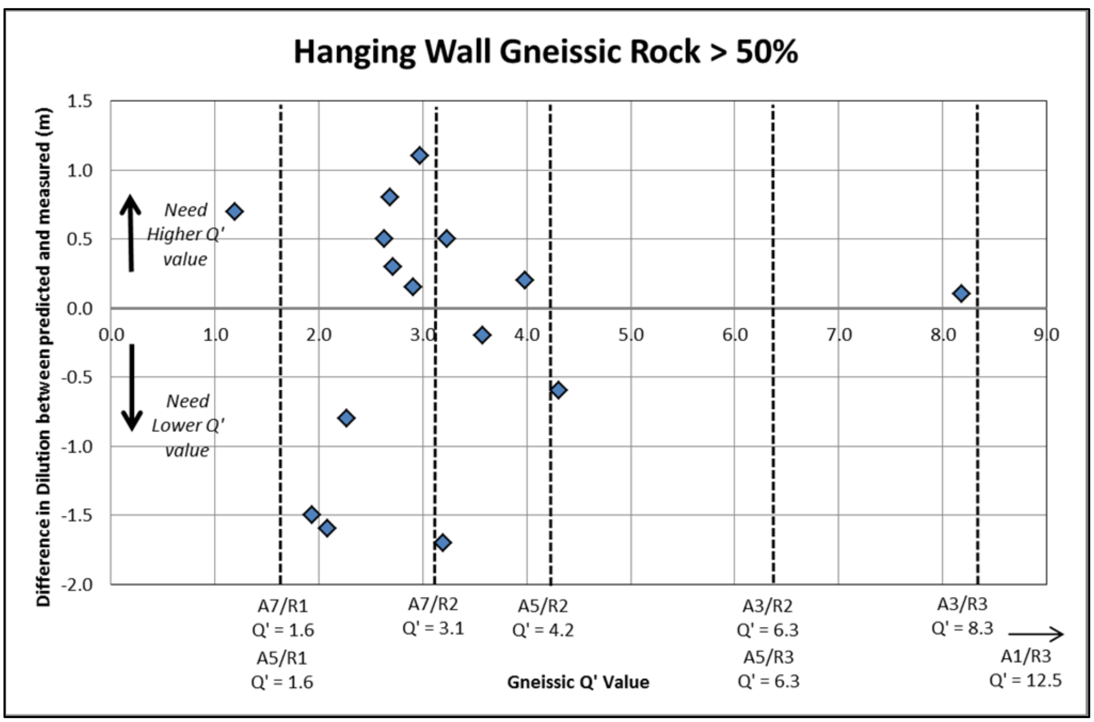


Figure 8-16 Difference between predicted and measured dilution versus Q' for stopes with more than 50% gneissic hanging wall rock.

8.3 Summary

The Parameter Based Q' approach is an improvement on the Sutton Q' and RMR_{76} to Q' approaches. The Parameter Based Q' approach is promising, and with additional data collection, further refinement could be possible. The link between the Q' joint alteration, J_a, and joint roughness, J_r, parameters and the RMR_{76} joint description is subjective and could be adjusted to improve results in the pegmatoidal rock types. For instance, the reduced number of joint sets used for the altered and weak pegmatoidal rocks may not have been warranted because the lowered Q' and dilution results suggest that a higher Q' would actually lead to better dilution predictions. Further adjustments to the Parameter Based Q' correlation, followed by additional stope case histories, is an area where further research could be conducted. Detailed core logging of joint surfaces, specifically for joint conditions, could help increase the confidence in the J_a and J_r parameters.

Chapter 9 – Conclusions and Recommendations

9.1 Summary and Assessment of Findings

Rock mass classification is important for the prediction of the engineering behaviour of a rock mass, including large underground openings such as stopes. Several different rock mass classification systems are used at the Cameco Corporation Eagle Point Mine. Not all of these systems are designed for engineering applications. This thesis has focused on applying varied sources of data describing the rock mass at the Eagle Point Mine for the prediction of dilution in open stopes.

The causes of dilution are varied, and of those causes, the stope geometry and local geological conditions were analyzed and summarized (Forster et al., 2007). Some dilution may be planned for a stope block so as to recover more ore material, but excessive dilution has a negative impact on the economic value of the stope. Better dilution prediction would allow for an appropriate strike length at the design phase, blast holes to be designed in the optimal location, and ground support to be installed in the proper pattern and location.

There are several key areas of research that were conducted:

1. Data collection and analysis to determine a correlation between intact rock strength, rock mass alteration, and rock type.
2. Parameter based correlation between classification systems.
3. Standardized assessment of rock mass conditions.
4. Standardized assessment of stope geometry and dilution.

9.1.1 **Comparison between Rock Strength and Rock Alteration Assessment**

The uranium deposits at the Eagle Point Mine are associated with varying degrees of hydrothermal alteration. The Geology staff at the mine use a site specific classification system based on the degree of alteration and the rock strength. These alteration (A) and strength (R) observations for certain rock types can be related to engineering properties. An attempt was made to compare the rock strength from point load testing to the degree of alteration (Forster et al., 2006) to remove the user subjectivity from the method. The point load tests indicated a general decrease in the maximum unconfined compressive strength of the five rock types assessed as the degree of alteration increased. There was significant scatter in the data and this approach was not pursued.

9.1.2 **Parameter Based Correlation between Classification Systems**

Previous work has been done relating the geological A/R values to Rock Quality Index, Q, values (Sutton, 1998) due to the extensive geological mapping values throughout the mine. This correlation was based on limited data and has been updated based on additional data and a new approach for correlating individual rock mass properties between classification systems. Rock Mass Rating (1976), RMR_{76}, values have been analyzed and included in the refinement of the A/R to Q' values for each rock type (Forster et al., 2012). A new method of comparing RMR_{76} observations and Q' ratings was proposed using the individual parameters.

9.1.3 **Standardized Assessment of Rock Mass Conditions**

The mine's previous geological A/R to Q' assumptions, the RMR_{76} to Q' equation based conversion values, and the Parameter Based Q' values were used for stope dilution estimates, and were then compared to the actual dilution from the stope performance. The Parameter Based Q' values are an improvement on the Sutton Q' values and RMR_{76} to Q' equation based conversion values, but there is still the opportunity for further refinement of the input data. A methodology was developed to estimate average rock mass properties based on overcut and undercut mapping as well as diamond drill hole data.

9.1.4 **Standardized Assessment of Stope Geometry and Dilution**

A process for post-production stope dilution analysis was developed (Forster, 2011) and has been applied to assess 27 case studies. The method of stope dilution planning and stope reconciliation for the Eagle Point Mine was previously highly subjective, and relied heavily on experience. A standardized and repeatable methodology for dilution planning and reconciliation was developed. The Parameter Based Q' correlation, tempered with geological understanding of site conditions and the A/R system, is an improvement on the mine's previous rock mass characterization for stope dilution prediction.

9.2 Recommendations for Future Research

Based on the results of this project, there are several recommendations for further research. The first is the establishment of a comprehensive database with further observations on rock mass properties within the ore zones, for further refinement of the data on rock mass classification.

One observation in particular which is difficult to obtain is the joint surface condition. Other properties may be visually estimated from a safe location underground, but the low strength and high alteration rock categories are typically shotcreted as soon as possible. They are not accessible for close inspection after the ground has been opened due to safety concerns, and after the ground is supported, they are covered with shotcrete. This is typically not a problem with the higher strength and lower alteration rock types, as they typically do not require shotcrete for support. They are also not usually found in the immediate hanging wall of stopes.

One comparison which may be possible is to calibrate the rock mass characterization with core logging data and drift mapping of the same location.

The Q' system is difficult to use in very poor rock masses such as are found at the Eagle Point Mine. All three approaches used for dilution prediction were poor at predicting stope dilution for the weaker ($Q' \leq 2.5$) pegmatoidal rock types. This may be due to the weak pegmatoidal rocks behaving more like a soil than a rock mass. Further investigation of the rock mass in terms of excavation stability of soils is recommended.

The number of joint sets parameter, J_n, was assumed to be less in the strongly altered and weaker pegmatoidal rock masses. This assumption should be examined further, as the J_n value intuitively should be higher for the strongly altered and weak pegmatoidal rock types.

Further case studies of mined stopes should be analyzed to be included in the analysis results as a method of continually improving the correlation.

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Appendix A

Point Load Test Results

Parameters
Core Diameter: 39 mm
F: 0.894

Test Number ("Hole"-"Test")	Segment Type	Position (m)	Test Result (kPa)	I_s	$I_{s\,50}$	UCS ($24*I_{s50}$)	Lithology	Min1	%	Min2	%	Min3	%	Total	Alteration	Comments
1817	FW	101.5	3680	2.340	2.092	50.2	HPGD	ill	50	clay	5			55	Moderate	
1817	FW	102	1540	0.979	0.876	21.0	HPGD	ill	50	clay	5			55	Moderate	
1817	FW	102.9	440	0.280	0.250	6.0	HPGD	ill	50	clay	5			55	Moderate	
1817-04	ORE	103.6	1230	0.782	0.699	16.8	HPGD	ill	50	clay	5			55	Moderate	
1817-13	HW	95.3	1000	0.636	0.569	13.6	BQFG	ill	50	clay	5			55	Moderate	
1817-14	HW	94.7	1800	1.144	1.023	24.6	BQFG	ill	50	clay	5			55	Moderate	
1817-15	HW	94.3	3440	2.187	1.956	46.9	BQFG	ill	50	clay	5			55	Moderate	
1817-16	HW	94.1	2700	1.717	1.535	36.8	BQFG	ill	50	clay	5			55	Moderate	
1817-17	HW	93.5	2800	1.780	1.592	38.2	BQFG	ill	50	clay	5			55	Moderate	
1817-18	HW	92.4	2320	1.475	1.319	31.7	BQFG	ill	50	clay	5			55	Moderate	
1817-19	HW	90.8	3400	2.162	1.933	46.4	BQFG	ill	50	clay	5			55	Moderate	
1817-20	HW	89	3700	2.352	2.103	50.5	BQFG	ill	50	clay	5			55	Moderate	
1819-01	FW	92.3	7800	4.959	4.434	106.4	BQFG	sil	20	hem	5	chl	10	35	Weak	
1819-02	FW	91.6	10320	6.561	5.867	140.8	BQFG	sil	20	hem	5	chl	10	35	Weak	
1819-03	FW	91	9120	5.798	5.185	124.4	HPGD	sil	40	chl	15			55	Moderate	
1819-04	ORE	89.5	1860	1.183	1.057	25.4	HPGD	sil	40	chl	15			55	Moderate	
1819-05	ORE	89	6940	4.412	3.945	94.7	HPGD	sil	40	chl	15			55	Moderate	
1819-06	ORE	88.5	4680	2.975	2.661	63.9	BQFG	sil	40	chl	15			55	Moderate	
1819-07	ORE	88	1700	1.081	0.966	23.2	BQFG	sil	40	chl	15			55	Moderate	
1819-12	ORE	85.5	1740	1.106	0.989	23.7	BQFG	chl	20	ill	20			40	Weak	
1819-14	HW	84.5	1720	1.094	0.978	23.5	BQFG	chl	20	ill	20			40	Weak	
1819-15	HW	84	1720	1.094	0.978	23.5	BQFG	chl	20	ill	20			40	Weak	
1819-16	HW	83.5	6060	3.853	3.445	82.7	BQFG	chl	15	hem	3	lim	1	19	Weak	
1819-18	HW	80	2280	1.450	1.296	31.1	BQFG	chl	15	hem	3	lim	1	19	Weak	
1819-19	HW	78.5	3240	2.060	1.842	44.2	BQFG	chl	15	hem	3	lim	1	19	Weak	
1819-20	HW	77.8	4000	2.543	2.274	54.6	BQFG	chl	15	hem	3	lim	1	19	Weak	
1821-01	FW	85	1920	1.221	1.092	26.2	BQFG	chl	20	hem	3			23	Weak	
1821-02	FW	84.5	3760	2.390	2.138	51.3	BQFG	chl	20	hem	3			23	Weak	
1821-03	FW	83.8	400	0.254	0.227	5.5	BQFG	chl	20	hem	3			23	Weak	
1821-04	ORE	83	5360	3.408	3.047	73.1	BQFG	ill	20	sil	15	hem	2	37	Weak	
1821-05	ORE	82.5	4900	3.115	2.786	66.9	BQFG	ill	20	sil	15	hem	2	37	Weak	
1821-11	ORE	79.5	5700	3.624	3.241	77.8	HPGD	ill	20	sil	15	hem	2	37	Weak	
1821-12	ORE	79	8100	5.150	4.605	110.5	HPGD	ill	20	sil	15	hem	2	37	Weak	
1821-13	HW	78.5	700	0.445	0.398	9.6	FXPR	ill	20	sil	15	hem	2	37	Weak	
1821-14	HW	78	980	0.623	0.557	13.4	FXPR	ill	20	sil	15	hem	2	37	Weak	
1821-15	HW	77.5	3160	2.009	1.796	43.1	FXPR	ill	20	sil	15	hem	2	37	Weak	
1821-16	HW	77	5260	3.344	2.990	71.8	FXPR	ill	20	sil	15	hem	2	37	Weak	
1821-17	HW	76	9000	5.722	5.117	122.8	FXPR	ill	20	sil	15	hem	2	37	Weak	
1821-18	HW	75	6600	4.196	3.752	90.1	BQFG	ill	20	sil	15	hem	2	37	Weak	
1821-19	HW	72	5020	3.192	2.854	68.5	HPGD	ill	20	sil	15	hem	2	37	Weak	
1821-20	HW	70	4480	2.848	2.547	61.1	HPGD	chl	20	lim	1	hem	2	23	Weak	
1827-01	FW	65	2620	1.666	1.490	35.7	BQFG	ill	20	lim	15	hem	7	42	Weak	
1827-02	FW	64.5	2560	1.628	1.455	34.9	BQFG	ill	20	lim	15	hem	7	42	Weak	
1827-03	FW	64	2520	1.602	1.433	34.4	BQFG	ill	20	lim	15	hem	7	42	Weak	
1827-04	ORE	63.7	2800	1.780	1.592	38.2	BQFG	ill	20	lim	15	hem	7	42	Weak	
1827-05	ORE	63	1360	0.865	0.773	18.6	BQFG	ill	40	lim	10	hem	7	57	Moderate	
1827-06	ORE	62.5	900	0.572	0.512	12.3	BQFG	ill	40	lim	10	hem	7	57	Moderate	
1827-07	ORE	60.8	4640	2.950	2.638	63.3	BQFG	ill	40	lim	10	hem	7	57	Moderate	
1827-08	ORE	60.4	2560	1.628	1.455	34.9	BQFG	ill	40	lim	10	hem	7	57	Moderate	
1827-09	ORE	60	1600	1.017	0.910	21.8	BQFG	ill	40	lim	10	hem	7	57	Moderate	
1827-10	ORE	58.9	3000	1.907	1.706	40.9	FXPR	ill	40	lim	10	hem	7	57	Moderate	
1827-11	ORE	58.6	5860	3.726	3.331	80.0	FXPR	ill	40	lim	10	hem	7	57	Moderate	
1827-12	ORE	58.2	2720	1.729	1.546	37.1	FXPR	ill	40	lim	10	hem	7	57	Moderate	
1827-13	HW	57.8	6000	3.815	3.411	81.9	FXPR	ill	40	lim	10	hem	7	57	Moderate	
1827-14	HW	57	10640	6.765	6.049	145.2	FXPR	ill	40	lim	10	hem	7	57	Moderate	
1827-15	HW	56.5	9240	5.874	5.253	126.1	FXPR	ill	40	lim	10	hem	7	57	Moderate	
1827-16	HW	56	5200	3.306	2.956	71.0	HPGD	ill	40	lim	10	hem	7	57	Moderate	
1827-17	HW	55.5	4260	2.708	2.422	58.1	HPGD	ill	40	lim	10	hem	7	57	Moderate	
1827-18	HW	55	3820	2.429	2.172	52.1	HPGD	ill	40	lim	10	hem	7	57	Moderate	
1827-19	HW	52	3280	2.085	1.865	44.8	HPGD	ill	40	lim	10	hem	7	57	Moderate	
1827-20	HW	49	5200	3.306	2.956	71.0	HPGD	ill	40	lim	10	hem	7	57	Moderate	
1829-01	FW	65	3100	1.971	1.762	42.3	BQFG	chl	20					20	Weak	
1829-02	FW	64.5	5220	3.319	2.968	71.2	BQFG	chl	20					20	Weak	
1829-03	FW	64	3900	2.479	2.217	53.2	BQFG	chl	20					20	Weak	
1829-04	ORE	63.7	11160	7.095	6.345	152.3	BQFG	chl	20					20	Weak	
1829-05	ORE	63	2300	1.462	1.308	31.4	BQFG	hem	50	clay	20	lim	5	75	Moderate	
1829-06	ORE	62.5	1140	0.725	0.648	15.6	BQFG	hem	50	clay	20	lim	5	75	Moderate	
1829-10	ORE	58.9	6660	4.234	3.786	90.9	FXPR	ill	30	hem	10	lim	3	43	Weak	
1829-14	HW	57	5320	3.382	3.024	72.6	FXPR	ill	30	hem	10	lim	3	43	Weak	
1829-15	HW	56.5	7960	5.061	4.525	108.6	FXPR	ill	30	hem	10	lim	3	43	Weak	
1829-16	HW	56	8020	5.099	4.559	109.4	FXPR	ill	30	hem	10	lim	3	43	Weak	
1829-17	HW	55.5	8600	5.468	4.889	117.3	FXPR	ill	30	hem	10	lim	3	43	Weak	
1829-18	HW	55	5480	3.484	3.115	74.8	FXPR	ill	30	hem	10	lim	3	43	Weak	
1829-19	HW	52	2240	1.424	1.273	30.6	FXPR	ill	30	hem	10	lim	3	43	Weak	

Test Number ("Hole"-"Test")	Segment Type	Position (m)	Test Result (kPa)	I_s	$I_{s\,50}$	UCS ($24*I_{s50}$)	Lithology	Min1	%	Min2	%	Min3	%	Total	Alteration	Comments
1831-01	FW	65	10120	6.434	5.753	138.1	HPGD	hem	50	lim	30			80	Moderate	
1831-02	FW	64.5	3400	2.162	1.933	46.4	BQFG	hem	50	lim	30			80	Moderate	
1831-03	FW	64	1700	1.081	0.966	23.2	BQFG	hem	50	lim	30			80	Moderate	
1831-05	ORE	63	4780	3.039	2.717	65.2	BQFG	ill	55	lim	3			58	Moderate	
1831-07	ORE	60.8	1600	1.017	0.910	21.8	BQFG	ill	55	lim	3			58	Moderate	
1831-09	ORE	60	2360	1.500	1.342	32.2	FXPR	ill	55	lim	3			58	Moderate	
1831-10	ORE	58.9	6320	4.018	3.593	86.2	FXPR	ill	55	lim	3			58	Moderate	
1831-11	ORE	58.6	6240	3.967	3.548	85.1	FXPR	ill	55	lim	3			58	Moderate	
1831-12	ORE	58.2	5200	3.306	2.956	71.0	FXPR	ill	55	lim	3			58	Moderate	
1831-13	HW	57.8	5200	3.306	2.956	71.0	MIN	hem	60	lim	20			80	Moderate	
1831-14	HW	57	6520	4.145	3.707	89.0	FXPR	ill	50	hem	5	lim	2	57	Moderate	
1831-15	HW	56.5	3520	2.238	2.001	48.0	FXPR	ill	50	hem	5	lim	2	57	Moderate	
1831-16	HW	56	5040	3.204	2.865	68.8	FXPR	ill	50	hem	5	lim	2	57	Moderate	
1831-17	HW	55.5	4240	2.696	2.410	57.9	FXPR	ill	50	hem	5	lim	2	57	Moderate	
1831-18	HW	55	1280	0.814	0.728	17.5	FXPR	ill	50	hem	5	lim	2	57	Moderate	
1831-19	HW	52	3660	2.327	2.081	49.9	FXPR	lim	50	hem	25			75	Moderate	
1831-20	HW	49	3280	2.085	1.865	44.8	FXPR	lim	50	hem	25			75	Moderate	
1835-01	FW	101.5	3340	2.123	1.899	45.6	BQFG	chl	25	sil	10			35	Weak	
1835-02	FW	101	4140	2.632	2.354	56.5	BQFG	chl	25	sil	10			35	Weak	
1835-03	FW	100.3	4100	2.607	2.331	55.9	BQFG	chl	25	sil	10			35	Weak	
1835-11	ORE	95	2380	1.513	1.353	32.5	BQFG	chl	30	ser	7			37	Weak	
1835-12	ORE	94.5	1340	0.852	0.762	18.3	BQFG	chl	30	ser	7			37	Weak	
1835-13	HW	94	5220	3.319	2.968	71.2	BQFG	chl	30	ser	7			37	Weak	
1835-14	HW	93.6	3000	1.907	1.706	40.9	BQFG	chl	30	ser	7			37	Weak	
1835-15	HW	93.2	5040	3.204	2.865	68.8	BQFG	chl	30	ser	7			37	Weak	
1835-18	HW	90	7300	4.641	4.150	99.6	BQFG	chl	30					30	Weak	
1835-19	HW	88	6440	4.094	3.661	87.9	BQFG	chl	30					30	Weak	
1835-20	HW	85	4960	3.153	2.820	67.7	BQFG	chl	30					30	Weak	
1837-01	FW	92.2	1820	1.157	1.035	24.8	HPGD	sil	20	hem	3			23	Weak	
1837-02	FW	91.5	7580	4.819	4.309	103.4	BQFG	chl	15	ser	15	lim	2	32	Weak	
1837-05	ORE	90	2980	1.895	1.694	40.7	BQFG	chl	15	ser	15	lim	2	32	Weak	
1837-06	ORE	89.6	1420	0.903	0.807	19.4	BQFG	chl	15	ser	15	lim	2	32	Weak	
1837-07	ORE	87.9	960	0.610	0.546	13.1	BQFG	chl	15	ser	15	lim	2	32	Weak	
1837-08	ORE	87.5	3980	2.530	2.263	54.3	BQFG	chl	15	ser	15	lim	2	32	Weak	
1837-13	HW	85	3100	1.971	1.762	42.3	BQFG	chl	30					30	Weak	
1837-14	HW	84.6	1240	0.788	0.705	16.9	BQFG	chl	30					30	Weak	
1837-16	HW	83.8	5020	3.192	2.854	68.5	BQFG	chl	30					30	Weak	
1837-17	HW	83.4	2500	1.589	1.421	34.1	BQFG	chl	30					30	Weak	
1837-18	HW	80	9160	5.824	5.208	125.0	HPGD	saus	15	chl	10			25	Weak	
1837-19	HW	77.5	5140	3.268	2.922	70.1	HPGD	saus	15	chl	10			25	Weak	
1837-20	HW	75.5	14800	9.409	8.414	201.9	HPGD	saus	15	chl	10			25	Weak	
1839-01	FW	86.6	8520	5.417	4.844	116.2	FXPR	sil	15	py	5	chl	7	27	Weak	
1839-02	FW	86	10340	6.574	5.878	141.1	FXPR	sil	15	py	5	chl	7	27	Weak	
1839-03	FW	85.3	9480	6.027	5.389	129.3	FXPR	sil	15	py	5	chl	7	27	Weak	
1839-13	HW	78.2	8760	5.569	4.980	119.5	HPGD	saus	15	hem	10	lim	5	30	Weak	
1839-14	HW	77.9	5640	3.586	3.206	77.0	HPGD	saus	15	hem	10	lim	5	30	Weak	
1839-15	HW	77.5	5900	3.751	3.354	80.5	HPGD	saus	15	hem	10	lim	5	30	Weak	
1839-16	HW	76.8	6740	4.285	3.832	92.0	HPGD	saus	15	hem	10	lim	5	30	Weak	
1839-17	HW	76.4	9620	6.116	5.469	131.3	HPGD	saus	15	hem	10	lim	5	30	Weak	
1839-18	HW	76	9820	6.243	5.583	134.0	HPGD	saus	15	hem	10	lim	5	30	Weak	
1839-19	HW	72	10100	6.421	5.742	137.8	HPGD	saus	15	hem	10	lim	5	30	Weak	
1839-20	HW	68.4	11320	7.197	6.436	154.5	HPGD	saus	15	hem	10	lim	5	30	Weak	
1841-01	FW	77	3920	2.492	2.229	53.5	BQFG	chl	25	sil	15			40	Weak	
1841-02	FW	76.5	1120	0.712	0.637	15.3	BQFG	chl	25	sil	15			40	Weak	
1841-03	FW	76	2300	1.462	1.308	31.4	BQFG	chl	25	sil	15			40	Weak	
1841-04	ORE	75.3	1780	1.132	1.012	24.3	MIN	chl	30	clay	15			45	Weak	
1841-05	ORE	75	1720	1.094	0.978	23.5	MIN	hem	30	sil	15	clay	7	52	Moderate	
1841-07	ORE	74.4	1620	1.030	0.921	22.1	MIN	chl	25	clay	20	lim	5	50	Moderate	
1841-08	ORE	74	1380	0.877	0.785	18.8	MIN	chl	25	clay	20	lim	5	50	Moderate	
1841-09	ORE	73.6	4360	2.772	2.479	59.5	BQFG	hem	35	sil	15			50	Moderate	
1841-10	ORE	71.6	10400	6.612	5.913	141.9	FXPR	saus	15	chl	10			25	Weak	
1841-11	ORE	71.2	13720	8.723	7.800	187.2	FXPR	saus	15	chl	10			25	Weak	
1841-12	ORE	70.8	11360	7.222	6.458	155.0	FXPR	saus	15	chl	10			25	Weak	
1841-13	HW	70.5	9620	6.116	5.469	131.3	FXPR	saus	15	chl	10			25	Weak	
1841-14	HW	69.7	2080	1.322	1.183	28.4	FXPR	saus	15	chl	10			25	Weak	
1841-16	HW	68.9	2880	1.831	1.637	39.3	FXPR	saus	15	chl	10			25	Weak	
1841-17	HW	68.5	12940	8.227	7.357	176.6	FXPR	saus	15	chl	10			25	Weak	
1841-18	HW	65	12000	7.629	6.822	163.7	FXPR	hem	35	lim	7	sil	10	52	Moderate	
1841-19	HW	63	7760	4.934	4.412	105.9	FXPR	hem	35	lim	7	sil	10	52	Moderate	
1841-20	HW	61	9200	5.849	5.230	125.5	FXPR	saus	25	sil	15	py	1	41	Weak	
1843-01	FW	71	4380	2.785	2.490	59.8	FXPR	sil	15	chl	15	clay	3	33	Weak	
1843-02	FW	70.5	8000	5.086	4.548	109.2	FXPR	sil	15	chl	15	clay	3	33	Weak	
1843-03	FW	70	3340	2.123	1.899	45.6	FXPR	hem	55	lim	5			60	Moderate	
1843-04	ORE	68.8	2320	1.475	1.319	31.7	BQFG	sil	20	chl	15			35	Weak	
1843-05	ORE	68.4	3060	1.945	1.740	41.8	BQFG	sil	20	chl	15			35	Weak	
1843-06	ORE	68	1060	0.674	0.603	14.5	BQFG	sil	20	chl	15			35	Weak	
1843-07	ORE	67.1	7780	4.946	4.423	106.2	MIN	hem	45	clay	15			60	Moderate	
1843-08	ORE	66.7	5580	3.548	3.172	76.1	MIN	hem	46	clay	16			62	Moderate	
1843-09	ORE	66.3	2820	1.793	1.603	38.5	MIN	hem	47	clay	17			64	Moderate	

Test Number ("Hole"-"Test")	Segment Type	Position (m)	Test Result (kPa)	I_s	$I_{s\,50}$	UCS ($24*I_{s50}$)	Lithology	Min1	%	Min2	%	Min3	%	Total	Alteration	Comments
1843-10	ORE	65.4	4120	2.619	2.342	56.2	BQFG	sil	25	ep	20			45	Weak	
1843-11	ORE	65	6660	4.234	3.786	90.9	BQFG	sil	25	ep	20			45	Weak	
1843-12	ORE	64.6	5580	3.548	3.172	76.1	BQFG	sil	25	ep	20			45	Weak	
1843-13	HW	64.1	2480	1.577	1.410	33.8	BQFG	sil	25	ep	20			45	Weak	
1843-14	HW	63.7	5520	3.509	3.138	75.3	BQFG	sil	25	ep	20			45	Weak	
1843-15	HW	63.3	4100	2.607	2.331	55.9	BQFG	sil	25	ep	20			45	Weak	
1843-16	HW	62.9	4260	2.708	2.422	58.1	BQFG	hem	45	lim	3			48	Weak	
1843-17	HW	62.5	4240	2.696	2.410	57.9	BQFG	hem	45	lim	3			48	Weak	
1843-18	HW	62	3060	1.945	1.740	41.8	BQFG	hem	45	lim	3			48	Weak	
1843-19	HW	59	7520	4.781	4.275	102.6	BQFG	hem	45	lim	3			48	Weak	
1843-20	HW	56	11100	7.057	6.310	151.5	FXPR	saus	25	sil	25			50	Moderate	
1843-21	HW	57.5	5040	3.204	2.865	68.8	BQFG	sil	20	ep	5			25	Weak	
1843-22	HW	58	6980	4.438	3.968	95.2	BQFG	hem	45	lim	3			48	Weak	
1845-01	FW	68.5	7980	5.073	4.537	108.9	HPGD	chl	30					30	Weak	
1845-02	FW	67.5	5580	3.548	3.172	76.1	HPGD	chl	30					30	Weak	
1845-03	FW	67	7020	4.463	3.991	95.8	HPGD	chl	30					30	Weak	
1845-13	HW	62.3	3380	2.149	1.922	46.1	BQFG	sil	40	ep	10			50	Moderate	
1845-14	HW	61.9	1000	0.636	0.569	13.6	BQFG	sil	40	ep	10			50	Moderate	
1845-15	HW	61.5	7940	5.048	4.514	108.3	BQFG	hem	55	sil	25			80	Moderate	
1845-16	HW	61.1	2240	1.424	1.273	30.6	BQFG	hem	55	sil	25			80	Moderate	
1845-17	HW	60.7	9780	6.218	5.560	133.4	BQFG	hem	55	sil	25			80	Moderate	
1845-18	HW	58.5	16160	10.274	9.187	220.5	BQFG	sil	30	ep	7			37	Weak	
1845-19	HW	56.5	6040	3.840	3.434	82.4	BQFG	sil	30	ep	7			37	Weak	
1845-20	HW	54.5	6880	4.374	3.911	93.9	BQFG	sil	30	ep	7			37	Weak	
1847-01	FW	65.5	960	0.610	0.546	13.1	BQFG	sil	25	ep	7	clay	3	35	Weak	
1847-02	FW	65	460	0.292	0.262	6.3	BQFG	sil	25	ep	7	clay	3	35	Weak	
1847-03	FW	64.5	2380	1.513	1.353	32.5	MIN	sil	25	ep	7	clay	3	35	Weak	
1847-05	ORE	63.1	3980	2.530	2.263	54.3	MIN	hem	35	sil	15	clay	5	55	Moderate	
1847-06	ORE	62.7	2180	1.386	1.239	29.7	FXPR	chl	20	sil	25			45	Weak	
1847-07	ORE	60.8	5200	3.306	2.956	71.0	BQFG	chl	20	sil	25			45	Weak	
1847-08	ORE	60.4	760	0.483	0.432	10.4	MIN	hem	40	lim	5			45	Weak	
1847-09	ORE	60	6440	4.094	3.661	87.9	BQFG	hem	25	sil	20			45	Weak	
1847-10	ORE	59.2	8680	5.518	4.935	118.4	BQFG	hem	25	sil	20			45	Weak	
1847-11	ORE	58.8	4100	2.607	2.331	55.9	BQFG	hem	25	sil	20			45	Weak	
1847-12	ORE	58.4	5200	3.306	2.956	71.0	BQFG	hem	25	sil	20			45	Weak	
1847-13	HW	58	2060	1.310	1.171	28.1	BQFG	chl	20	sil	10	ep	5	35	Weak	
1847-14	HW	57.6	2760	1.755	1.569	37.7	BQFG	chl	20	sil	10	ep	5	35	Weak	
1847-15	HW	57.2	13620	8.659	7.743	185.8	BQFG	chl	20	sil	10	ep	5	35	Weak	
1847-16	HW	56.8	4880	3.103	2.774	66.6	BQFG	chl	20	sil	10	ep	5	35	Weak	
1847-17	HW	56.4	10000	6.358	5.685	136.4	BQFG	chl	20	sil	10	ep	5	35	Weak	
1847-18	HW	54	9460	6.014	5.378	129.1	FXPR	chl	15					15	Weak	
1847-19	HW	52	5640	3.586	3.206	77.0	FXPR	chl	15					15	Weak	
1847-20	HW	50	4520	2.874	2.570	61.7	FXPR	chl	15					15	Weak	
1787-01	FW	86	5960	3.789	3.388	81.3	FXPR	clay	10	lim	2			12	Fresh	
1787-02	FW	85.2	9880	6.281	5.617	134.8	FXPR	clay	10	lim	2			12	Fresh	
1787-03	FW	84.6	4620	2.937	2.627	63.0	FXPR	clay	10	lim	2			12	Fresh	
1787-07	ORE	82.4	6900	4.387	3.923	94.1	FXPR	lim	10	clay	10			20	Weak	
1787-08	ORE	82	6500	4.132	3.695	88.7	FXPR	lim	10	clay	10			20	Weak	
1787-09	ORE	81.6	4840	3.077	2.752	66.0	FXPR	lim	10	clay	10			20	Weak	
1787-13	HW	79	9400	5.976	5.344	128.3	FXPR	lim	10	clay	10			20	Weak	
1787-14	HW	78.6	10220	6.498	5.810	139.4	FXPR	lim	10	clay	10			20	Weak	
1787-15	HW	78.2	6360	4.043	3.616	86.8	FXPR	clay	20	lim	3	hem	2	25	Weak	
1787-16	HW	77.8	7120	4.527	4.048	97.1	FXPR	clay	20	lim	3	hem	2	25	Weak	
1787-17	HW	77.4	11000	6.993	6.254	150.1	FXPR	clay	20	lim	3	hem	2	25	Weak	
1787-18	HW	75	8280	5.264	4.707	113.0	FXPR	clay	20	lim	3	hem	2	25	Weak	
1787-19	HW	72	6580	4.183	3.741	89.8	FXPR	clay	20	lim	3	hem	2	25	Weak	
1787-20	HW	69	7900	5.023	4.491	107.8	FXPR	clay	20	lim	3	hem	2	25	Weak	
1797-01	FW	66	5240	3.331	2.979	71.5	FXPR	sil	50	ill	25			75	Moderate	
1797-02	FW	65.2	6860	4.361	3.900	93.6	FXPR	sil	50	ill	25			75	Moderate	
1797-03	FW	64.5	12180	7.744	6.924	166.2	FXPR	sil	50	ill	25			75	Moderate	
1797-04	ORE	64	5840	3.713	3.320	79.7	FXPR	sil	50	ill	25			75	Moderate	
1797-05	ORE	63.6	4520	2.874	2.570	61.7	MIN	sil	50	ill	25			75	Moderate	
1797-06	ORE	63.2	1400	0.890	0.796	19.1	MIN	sil	50	ill	25			75	Moderate	
1797-08	ORE	62	4400	2.797	2.501	60.0	BQFG	chl	20	ill	10			30	Weak	
1797-09	ORE	61.6	2620	1.666	1.490	35.7	BQFG	chl	20	ill	10			30	Weak	
1797-11	ORE	60.4	7280	4.628	4.139	99.3	BQFG	chl	20	ill	10			30	Weak	
1797-12	ORE	60	6340	4.031	3.604	86.5	BQFG	chl	20	ill	10			30	Weak	
1797-13	HW	59.6	11880	7.553	6.754	162.1	BQFG	chl	20					20	Weak	
1797-14	HW	59.2	10920	6.943	6.208	149.0	BQFG	chl	20					20	Weak	
1797-15	HW	58.8	10400	6.612	5.913	141.9	BQFG	chl	20					20	Weak	
1797-16	HW	58.4	13200	8.392	7.504	180.1	BQFG	chl	20					20	Weak	
1797-17	HW	58	1100	0.699	0.625	15.0	BQFG	chl	20					20	Weak	
1797-18	HW	55	12560	7.985	7.141	171.4	BQFG							0	Fresh	
1797-19	HW	53	21740	13.822	12.359	296.6	BQFG							0	Fresh	
1797-20	HW	50.5	19120	12.156	10.870	260.9	BQFG							0	Fresh	
1797-21	HW	50.2	15100	9.600	8.585	206.0	BQFG							0	Fresh	
1815-01	FW	71	5860	3.726	3.331	80.0	BQFG	ill	20	hem	5			25	Weak	
1815-02	FW	70.2	8240	5.239	4.685	112.4	BQFG	ill	20	hem	5			25	Weak	
1815-03	FW	69.4	7460	4.743	4.241	101.8	BQFG	ill	20	hem	5			25	Weak	

Test Number ("Hole"-"Test")	Segment Type	Position (m)	Test Result (kPa)	I_s	$I_{s\,50}$	UCS ($24*I_{s50}$)	Lithology	Min1	%	Min2	%	Min3	%	Total	Alteration	Comments
1815-04	ORE	68.7	5540	3.522	3.150	75.6	MIN	lim	20	ill	30			50	Moderate	
1815-05	ORE	68.3	1560	0.992	0.887	21.3	MIN	lim	20	ill	30			50	Moderate	
1815-06	ORE	67.9	4860	3.090	2.763	66.3	MIN	lim	20	ill	30			50	Moderate	
1815-07	ORE	66.9	4860	3.090	2.763	66.3	MIN	lim	20	ill	30			50	Moderate	
1815-08	ORE	66.5	3300	2.098	1.876	45.0	MIN	lim	20	ill	30			50	Moderate	
1815-09	ORE	66.1	4440	2.823	2.524	60.6	MIN	lim	20	ill	30			50	Moderate	
1815-10	ORE	65.2	4240	2.696	2.410	57.9	BQFG	ill	20					20	Weak	
1815-11	ORE	64.8	3420	2.174	1.944	46.7	BQFG	ill	20					20	Weak	
1815-12	ORE	64.4	7580	4.819	4.309	103.4	BQFG	ill	20					20	Weak	
1815-13	HW	64	2920	1.856	1.660	39.8	BQFG	ill	20					20	Weak	
1815-14	HW	63.6	10460	6.650	5.947	142.7	BQFG	ill	20					20	Weak	
1815-15	HW	63.2	11240	7.146	6.390	153.4	BQFG	ill	20					20	Weak	
1815-16	HW	62.8	13200	8.392	7.504	180.1	BQFG	ill	20					20	Weak	
1815-17	HW	62.4	3640	2.314	2.069	49.7	BQFG	ill	20					20	Weak	
1815-18	HW	60	5260	3.344	2.990	71.8	BQFG	chl	25	hem	5			30	Weak	
1815-19	HW	58	11080	7.044	6.299	151.2	BQFG	chl	25	hem	5			30	Weak	
1815-20	HW	55	12160	7.731	6.913	165.9	BQFG	chl	20	hem	3	lim	1	24	Weak	
1799-01	FW	98.5	2120	1.348	1.205	28.9	HPGD	sil	30					30	Weak	
1799-02	FW	97.8	4420	2.810	2.513	60.3	HPGD	sil	30					30	Weak	
1799-03	FW	97.1	940	0.598	0.534	12.8	HPGD	clay	60	lim	7			67	Moderate	
1799-14	HW	87.2	6060	3.853	3.445	82.7	BQFG	chl	25	hem	1			26	Weak	
1799-15	HW	86.8	9960	6.332	5.662	135.9	BQFG	chl	25	hem	1			26	Weak	
1799-16	HW	86.4	3800	2.416	2.160	51.8	BQFG	chl	25	hem	1			26	Weak	
1799-17	HW	86	12280	7.807	6.981	167.6	BQFG	chl	25	hem	1			26	Weak	
1799-18	HW	84	2220	1.411	1.262	30.3	BQFG	chl	25	hem	1			26	Weak	
1799-19	HW	81	5140	3.268	2.922	70.1	BQFG	chl	25	hem	1			26	Weak	
1799-20	HW	78.4	6260	3.980	3.559	85.4	BQFG	chl	20	sil	5	hem	7	32	Weak	
1801-01	FW	89	5760	3.662	3.275	78.6	QFGN	sil	25	hem	2			27	Weak	
1801-02	FW	88.6	6060	3.853	3.445	82.7	QFGN	sil	25	hem	2			27	Weak	
1801-03	FW	87.8	7500	4.768	4.264	102.3	QFGN	sil	25	hem	2			27	Weak	
1801-10	ORE	79.4	3380	2.149	1.922	46.1	QFGN	chl	15	lim	5	sil	7	27	Weak	
1801-13	HW	77.8	5060	3.217	2.877	69.0	QFGN	hem	20	saus	7	chl	7	34	Weak	
1801-14	HW	77.4	5000	3.179	2.843	68.2	BQFG	chl	15	hem	5			20	Weak	
1801-15	HW	77	17980	11.431	10.222	245.3	BQFG	chl	15	hem	5			20	Weak	
1801-16	HW	76.6	7800	4.959	4.434	106.4	BQFG	chl	5	saus	5			10	Fresh	
1801-17	HW	76.2	11520	7.324	6.549	157.2	HPGD	chl	5	saus	5			10	Fresh	
1801-18	HW	74.5	10480	6.663	5.958	143.0	QFGN	saus	5	chl	10			15	Weak	
1801-19	HW	71.5	15560	9.893	8.846	212.3	QFGN	saus	5	chl	10			15	Weak	
1801-20	HW	68.5	15480	9.842	8.801	211.2	BQFG	chl	20					20	Weak	
1803-01	FW	85	4040	2.568	2.297	55.1	BQFG	lim	15	sil	15			30	Weak	
1803-02	FW	83.9	7080	4.501	4.025	96.6	MIN	hem	60	lim	7			67	Moderate	
1803-03	FW	83.4	180	0.114	0.102	2.5	MIN	chl	20	clay	10			30	Weak	
1803-04	ORE	82.2	2200	1.399	1.251	30.0	MIN	chl	15	clay	10			25	Weak	
1803-05	ORE	81.8	3240	2.060	1.842	44.2	MIN	chl	15	clay	10			25	Weak	
1803-06	ORE	81.4	3600	2.289	2.047	49.1	FXPR	chl	25	clay	7			32	Weak	
1803-07	ORE	75.4	2480	1.577	1.410	33.8	FXPR	lim	7	ill	7			14	Fresh	
1803-08	ORE	75	4320	2.747	2.456	58.9	FXPR	lim	7	ill	7			14	Fresh	
1803-09	ORE	74.6	4300	2.734	2.445	58.7	FXPR	lim	7	ill	7			14	Fresh	
1803-10	ORE	72.2	2540	1.615	1.444	34.7	FLT	clay	15					15	Weak	
1803-11	ORE	71.8	3560	2.263	2.024	48.6	FLT	sil	20					20	Weak	
1803-12	ORE	71.4	5580	3.548	3.172	76.1	FLT	sil	20					20	Weak	
1803-13	HW	71	4200	2.670	2.388	57.3	FXPR	chl	15	sil	15			30	Weak	
1803-14	HW	69.6	7180	4.565	4.082	98.0	FXPR	chl	15	sil	15			30	Weak	
1803-15	HW	69.2	8860	5.633	5.037	120.9	FXPR	chl	15	sil	15			30	Weak	
1803-16	HW	68.8	7660	4.870	4.355	104.5	FXPR	chl	15	sil	15			30	Weak	
1803-17	HW	68.4	8740	5.557	4.969	119.3	FXPR	chl	15	sil	15			30	Weak	
1803-18	HW	66.5	5120	3.255	2.911	69.9	FLT	sil	20	chl	10	py	1	31	Weak	
1803-19	HW	64	4140	2.632	2.354	56.5	FLT	sil	20	chl	10	py	1	31	Weak	
1803-20	HW	61.5	13060	8.303	7.425	178.2	FXPR	chl	15					15	Weak	
1803-21	HW	50	11680	7.426	6.640	159.4	BQFG	chl	25	sil	10			35	Weak	
1805-01	FW	78.8	3400	2.162	1.933	46.4	FXPR	sil	30	ep	5			35	Weak	
1805-03	FW	77.2	1620	1.030	0.921	22.1	FXPR	sil	25	chl	15			40	Weak	
1805-04	ORE	76.8	3840	2.441	2.183	52.4	FXPR	sil	25	chl	15			40	Weak	
1805-05	ORE	76.4	3380	2.149	1.922	46.1	FXPR	sil	25	chl	15			40	Weak	
1805-07	ORE	71.4	6040	3.840	3.434	82.4	QFGN	sil	25					25	Weak	
1805-08	ORE	71	5020	3.192	2.854	68.5	QFGN	sil	25					25	Weak	
1805-09	ORE	70.6	3200	2.034	1.819	43.7	QFGN	sil	25					25	Weak	
1805-13	HW	65.5	9700	6.167	5.515	132.3	QFGN	hem	20	sil	15			35	Weak	
1805-14	HW	65.1	13260	8.430	7.538	180.9	QFGN	hem	20	sil	15			35	Weak	
1805-15	HW	64.7	12240	7.782	6.959	167.0	QFGN	sil	15	ser	7	ep	7	29	Weak	
1805-16	HW	64.3	10920	6.943	6.208	149.0	QFGN	sil	15	ser	7	ep	7	29	Weak	
1805-17	HW	63.9	1620	1.030	0.921	22.1	BQFG	sil	15	ser	7	ep	7	29	Weak	
1805-18	HW	62.5	9380	5.963	5.333	128.0	BQFG	sil	15	ser	7	ep	7	29	Weak	
1805-19	HW	59	4760	3.026	2.706	64.9	FLT	hem	15	chl	30			45	Weak	
1805-20	HW	56.5	6140	3.904	3.491	83.8	FLT	hem	15	chl	30			45	Weak	
1807-01	FW	71.8	3200	2.034	1.819	43.7	HPGD	hem	10	lim	10			20	Weak	
1807-02	FW	71	7520	4.781	4.275	102.6	FLT	sil	40	ep	7			47	Weak	
1807-03	FW	70.3	5260	3.344	2.990	71.8	FLT	sil	40	ep	7			47	Weak	
1807-04	ORE	69.9	4200	2.670	2.388	57.3	FLT	sil	40	ep	7			47	Weak	

Test Number ("Hole"-"Test")	Segment Type	Position (m)	Test Result (kPa)	I_s	$I_{s\,50}$	UCS (24*I_{s50})	Lithology	Min1	%	Min2	%	Min3	%	Total	Alteration	Comments
1807-05	ORE	69.5	9540	6.065	5.424	130.2	FLT	sil	40	ep	7			47	Weak	
1807-06	ORE	69.1	3200	2.034	1.819	43.7	MIN	sil	40	ep	7			47	Weak	
1807-07	ORE	67.4	3680	2.340	2.092	50.2	MIN	sil	30	lim	7			37	Weak	
1807-08	ORE	67	2440	1.551	1.387	33.3	MIN	sil	30	lim	7			37	Weak	
1807-09	ORE	66.6	3440	2.187	1.956	46.9	MIN	sil	30	lim	7			37	Weak	
1807-10	ORE	64.8	10360	6.587	5.890	141.4	QFGN	saus	25	sil	15			40	Weak	
1807-11	ORE	64.4	6200	3.942	3.525	84.6	QFGN	saus	25	sil	15			40	Weak	
1807-12	ORE	64	7960	5.061	4.525	108.6	QFGN	saus	25	sil	15			40	Weak	
1807-13	HW	63.5	3300	2.098	1.876	45.0	BQFG	saus	25	sil	15			40	Weak	
1807-14	HW	63.1	5460	3.471	3.104	74.5	BQFG	lim	15	sil	25			40	Weak	
1807-15	HW	62.7	4720	3.001	2.683	64.4	MIN	lim	15	sil	25			40	Weak	
1807-16	HW	62.3	1540	0.979	0.876	21.0	FXPR	hem	40	clay	7	sil	15	62	Moderate	
1807-17	HW	61.9	5620	3.573	3.195	76.7	FXPR	hem	40	clay	7	sil	15	62	Moderate	
1807-18	HW	60	9820	6.243	5.583	134.0	FXPR	hem	40	clay	7	sil	15	62	Moderate	
1807-19	HW	57	2160	1.373	1.228	29.5	QFGN	hem	40	clay	7	sil	15	62	Moderate	
1807-20	HW	54.5	7180	4.565	4.082	98.0	HPGD	saus	25					25	Weak	
1807-21	HW	56	3840	2.441	2.183	52.4	QFGN	hem	55	sil	20			75	Moderate	
1807-22	HW	58	6020	3.827	3.422	82.1	BQFG	sil	30	lim	5			35	Weak	
1809-01	FW	65.4	4720	3.001	2.683	64.4	BQFG	sil	15	ep	15	ser	5	35	Weak	
1809-02	FW	64.8	1480	0.941	0.841	20.2	BQFG	sil	15	ep	15	ser	5	35	Weak	
1809-10	ORE	59.5	13360	8.494	7.595	182.3	BQFG	sil	15					15	Weak	
1809-11	ORE	59.1	13020	8.278	7.402	177.6	BQFG	sil	15					15	Weak	
1809-12	ORE	58.7	6600	4.196	3.752	90.1	BQFG	sil	15					15	Weak	
1809-13	HW	58.4	13920	8.850	7.914	189.9	BQFG	sil	15					15	Weak	
1809-14	HW	58	14900	9.473	8.471	203.3	BQFG	sil	15					15	Weak	
1809-15	HW	57.6	12220	7.769	6.947	166.7	BQFG	sil	15					15	Weak	
1809-16	HW	57.2	15660	9.956	8.903	213.7	BQFG	sil	15					15	Weak	
1809-17	HW	56.8	10020	6.370	5.696	136.7	BQFG	sil	15					15	Weak	
1809-19	HW	49.5	14380	9.142	8.175	196.2	BQFG	hem	45					45	Weak	
1809-20	HW	48.9	8940	5.684	5.082	122.0	BQFG	hem	45					45	Weak	
1811-01	FW	65	3460	2.200	1.967	47.2	QFGN	ill	50	lim	2	hem	3	55	Moderate	
1811-02	FW	64.2	3760	2.390	2.138	51.3	MIN	ill	50	lim	2	hem	3	55	Moderate	
1811-03	FW	63.4	4220	2.683	2.399	57.6	QFGN	ill	50	lim	2	hem	3	55	Moderate	
1811-04	ORE	63	8820	5.607	5.014	120.3	QFGN	ill	50	lim	2	hem	3	55	Moderate	
1811-05	ORE	62.6	5580	3.548	3.172	76.1	FXPR	ill	50	lim	2	hem	3	55	Moderate	
1811-06	ORE	62.2	6700	4.260	3.809	91.4	FXPR	ill	2					2	Fresh	
1811-07	ORE	60.4	7820	4.972	4.446	106.7	FXPR	ill	2					2	Fresh	
1811-08	ORE	60	13660	8.685	7.766	186.4	FXPR	ill	2					2	Fresh	
1811-09	ORE	59.6	15300	9.727	8.698	208.8	FXPR	ill	2					2	Fresh	
1811-10	ORE	58.1	6660	4.234	3.786	90.9	BQFG	ill	25	lim	7			32	Weak	
1811-11	ORE	57.7	4980	3.166	2.831	67.9	BQFG	ill	25	lim	7			32	Weak	
1811-12	ORE	57.3	860	0.547	0.489	11.7	BQFG	ill	25	lim	7			32	Weak	
1811-13	HW	57	3200	2.034	1.819	43.7	BQFG	ill	25	lim	7			32	Weak	
1811-14	HW	56.6	2420	1.539	1.376	33.0	BQFG	ill	25	lim	7			32	Weak	
1811-15	HW	56.2	4960	3.153	2.820	67.7	BQFG	ill	25	lim	7			32	Weak	
1811-16	HW	55.8	9020	5.735	5.128	123.1	BQFG	ill	25	lim	7			32	Weak	
1811-17	HW	55.4	20080	12.766	11.416	274.0	BQFG	chl	5					5	Fresh	
1811-18	HW	52.5	9580	6.091	5.446	130.7	BQFG	ill	40	sil	10	lim	1	51	Moderate	
1811-19	HW	50	3980	2.530	2.263	54.3	BQFG	ill	15	hem	3			18	Weak	
1811-20	HW	47.5	3960	2.518	2.251	54.0	BQFG	ill	15	hem	3			18	Weak	
1813-01	FW	75.5	12560	7.985	7.141	171.4	QFGN	ill	50					50	Moderate	
1813-02	FW	74.7	11740	7.464	6.674	160.2	QFGN	ill	50					50	Moderate	
1813-03	FW	73.9	3540	2.251	2.013	48.3	QFGN	lim	50	hem	20			70	Moderate	
1813-04	ORE	73.5	5980	3.802	3.400	81.6	QFGN	lim	50	hem	20			70	Moderate	
1813-05	ORE	73.1	8140	5.175	4.628	111.1	QFGN	lim	50	hem	20			70	Moderate	
1813-06	ORE	72.7	3220	2.047	1.831	43.9	QFGN	lim	50	hem	20			70	Moderate	
1813-07	ORE	70.4	3300	2.098	1.876	45.0	FXPR	lim	50	hem	20			70	Moderate	
1813-08	ORE	70	3120	1.984	1.774	42.6	FXPR	ill	20	lim	5			25	Weak	
1813-09	ORE	69.6	3340	2.123	1.899	45.6	QFGN	ill	20	lim	5			25	Weak	
1813-10	ORE	68.6	4140	2.632	2.354	56.5	QFGN	ill	20	lim	5			25	Weak	
1813-11	ORE	68.2	3360	2.136	1.910	45.8	QFGN	ill	20	lim	5			25	Weak	
1813-12	ORE	67.8	9980	6.345	5.674	136.2	QFGN	ill	20	lim	5			25	Weak	
1813-13	HW	67.4	8740	5.557	4.969	119.3	QFGN	ill	20	lim	5			25	Weak	
1813-15	HW	66.8	3460	2.200	1.967	47.2	FXPR	hem	60	lim	5			65	Moderate	
1813-16	HW	66.4	3080	1.958	1.751	42.0	FXPR	hem	60	lim	5			65	Moderate	
1813-17	HW	66	5440	3.459	3.093	74.2	FXPR	hem	60	lim	5			65	Moderate	
1813-18	HW	63.5	10540	6.701	5.992	143.8	FXPR	chl	15	hem	3			18	Weak	
1813-19	HW	61.5	18360	11.673	10.438	250.5	FXPR	chl	15	hem	3			18	Weak	
1813-20	HW	59	15120	9.613	8.596	206.3	FXPR	chl	15	hem	3			18	Weak	
1905-01	FW	101.4	1560	0.992	0.887	21.3	HPGD	clay	50	hem	5	lim	2	57	Moderate	
1905-02	FW	100.9	2360	1.500	1.342	32.2	HPGD	clay	50	hem	5	lim	2	57	Moderate	
1905-03	FW	100.4	3780	2.403	2.149	51.6	HPGD	clay	50	hem	5	lim	2	57	Moderate	
1905-07	ORE	93.1	6640	4.221	3.775	90.6	HPGD	hem	40	lim	20			60	Moderate	
1905-08	ORE	92.5	5680	3.611	3.229	77.5	HPGD	hem	40	lim	20			60	Moderate	
1905-09	ORE	91.9	5660	3.598	3.218	77.2	HPGD	hem	40	lim	20			60	Moderate	
1905-13	HW	84.6	2340	1.488	1.330	31.9	HPGD	ill	20	hem	3	sil	10	33	Weak	
1905-14	HW	84.2	2580	1.640	1.467	35.2	HPGD	ill	20	hem	3	sil	10	33	Weak	
1905-15	HW	83.8	5060	3.217	2.877	69.0	HPGD	ill	20	hem	3	sil	10	33	Weak	
1905-16	HW	83.4	8180	5.201	4.650	111.6	HPGD	ill	20	hem	3	sil	10	33	Weak	
1905-17	HW	83	10000	6.358	5.685	136.4	HPGD	ill	20	hem	3	sil	10	33	Weak	

Test Number ("Hole"-"Test")	Segment Type	Position (m)	Test Result (kPa)	I_s	$I_{s\,50}$	UCS (24*I_{s50})	Lithology	Min1	%	Min2	%	Min3	%	Total	Alteration	Comments
1905-21	HW	71.8	7100	4.514	4.036	96.9	HPGD	clay	40	lim	2			42	Weak	
1905-22	HW	71.3	12160	7.731	6.913	165.9	HPGD	clay	40	lim	2			42	Weak	
1905-23	HW	70.8	3100	1.971	1.762	42.3	HPGD	clay	40	lim	2			42	Weak	
1905-24	HW	70.3	14580	9.269	8.289	198.9	BQFG	chl	10					10	Fresh	
1905-25	HW	69.8	15520	9.867	8.823	211.8	BQFG	chl	10					10	Fresh	
1905-26	HW	69.3	17520	11.139	9.960	239.0	BQFG	chl	10					10	Fresh	
1905-27	HW	68.8	4980	3.166	2.831	67.9	BQFG	chl	10					10	Fresh	
1905-28	HW	67.5	9940	6.320	5.651	135.6	BQFG	chl	10					10	Fresh	
1907-01	FW	84.8	6500	4.132	3.695	88.7	HPGD	chl	20	lim	5	ep	2	27	Weak	
1907-02	FW	84.2	3600	2.289	2.047	49.1	HPGD	chl	20	lim	5	ep	2	27	Weak	
1907-03	FW	83.6	2120	1.348	1.205	28.9	HPGD	ill	20	hem	7	lim	5	32	Weak	
1907-05	ORE	82.2	1440	0.916	0.819	19.6	MIN	ill	20	hem	7	lim	5	32	Weak	
1907-06	ORE	81.8	3740	2.378	2.126	51.0	MIN	ill	20	hem	7	lim	5	32	Weak	
1907-07	ORE	78.4	5180	3.293	2.945	70.7	MIN	ill	20	clay	20			40	Weak	
1907-08	ORE	78	1960	1.246	1.114	26.7	MIN	ill	20	clay	20			40	Weak	
1907-09	ORE	77.6	2600	1.653	1.478	35.5	MIN	ill	20	clay	20			40	Weak	
1907-10	ORE	75.3	3580	2.276	2.035	48.8	MIN	hem	50	lim	5			55	Moderate	
1907-11	ORE	74.9	3960	2.518	2.251	54.0	MIN	hem	50	lim	5			55	Moderate	
1907-12	ORE	74.5	6880	4.374	3.911	93.9	BQFG	chl	20	sil	1			21	Weak	
1907-13	HW	74.2	11300	7.184	6.424	154.2	BQFG	chl	20	sil	1			21	Weak	
1907-14	HW	73.8	8920	5.671	5.071	121.7	BQFG	chl	20	sil	1			21	Weak	
1907-15	HW	73.4	8400	5.340	4.776	114.6	BQFG	chl	20	sil	1			21	Weak	
1907-16	HW	73	7560	4.806	4.298	103.2	BQFG	chl	20	sil	1			21	Weak	
1907-17	HW	72.6	5020	3.192	2.854	68.5	BQFG	chl	20	sil	1			21	Weak	
1907-18	HW	70.2	5500	3.497	3.127	75.0	BQFG	chl	20	sil	1			21	Weak	
1909-01	FW	77.5	8100	5.150	4.605	110.5	FXPR	clay	15	hem	10	lim	5	30	Weak	
1909-02	FW	76.8	3100	1.971	1.762	42.3	FXPR	clay	15	hem	10	lim	5	30	Weak	
1909-03	FW	76.1	3560	2.263	2.024	48.6	FXPR	clay	15	hem	10	lim	5	30	Weak	
1909-04	ORE	75.5	2520	1.602	1.433	34.4	FXPR	clay	15	hem	10	lim	5	30	Weak	
1909-05	ORE	75.1	5400	3.433	3.070	73.7	MIN	clay	15	hem	10	lim	5	30	Weak	
1909-06	ORE	74.7	4320	2.747	2.456	58.9	MIN	clay	15	hem	10	lim	5	30	Weak	
1909-07	ORE	72.9	3660	2.327	2.081	49.9	MIN	clay	15	hem	10	lim	5	30	Weak	
1909-08	ORE	72.5	2300	1.462	1.308	31.4	MIN	clay	15	hem	10	lim	5	30	Weak	
1909-09	ORE	72.1	3400	2.162	1.933	46.4	MIN	clay	15	hem	10	lim	5	30	Weak	
1909-10	ORE	70.8	4740	3.014	2.695	64.7	FXPR	clay	15	hem	10	lim	5	30	Weak	
1909-11	ORE	70.4	8980	5.709	5.105	122.5	FXPR	clay	15	hem	10	lim	5	30	Weak	
1909-12	ORE	70	5140	3.268	2.922	70.1	FXPR	clay	15	hem	10	lim	5	30	Weak	
1909-13	HW	69.6	2740	1.742	1.558	37.4	FXPR	clay	15	hem	10	lim	5	30	Weak	
1909-14	HW	69.2	3400	2.162	1.933	46.4	FXPR	clay	15	hem	10	lim	5	30	Weak	
1909-15	HW	68.8	3980	2.530	2.263	54.3	FXPR	clay	15	hem	10	lim	5	30	Weak	
1909-16	HW	68.4	3600	2.289	2.047	49.1	FXPR	clay	15	hem	10	lim	5	30	Weak	
1909-17	HW	68	4420	2.810	2.513	60.3	FXPR	clay	15	hem	10	lim	5	30	Weak	
1909-18	HW	67	3180	2.022	1.808	43.4	FXPR	clay	15	hem	10	lim	5	30	Weak	
1909-19	HW	64	1800	1.144	1.023	24.6	FXPR	clay	15	hem	10	lim	5	30	Weak	
1909-20	HW	61	22140	14.076	12.587	302.1	FXPR	chl	15	sil	10	ep	3	28	Weak	
1909-21	HW	60.4	13400	8.519	7.618	182.8	FXPR	chl	15	sil	10	ep	3	28	Weak	
1911-01	FW	88.4	10720	6.815	6.094	146.3	FXPR	ill	25					25	Weak	
1911-02	FW	87.6	1200	0.763	0.682	16.4	HPGD	ill	25	hem	7			32	Weak	
1911-03	FW	86.8	5040	3.204	2.865	68.8	HPGD	ill	25	hem	7			32	Weak	
1911-13	HW	58	3160	2.009	1.796	43.1	FXPR	hem	60	lim	2			62	Moderate	
1911-14	HW	57.6	5180	3.293	2.945	70.7	FXPR	hem	60	lim	2			62	Moderate	
1911-15	HW	57.2	5060	3.217	2.877	69.0	FXPR	hem	60	lim	2			62	Moderate	
1911-16	HW	56.8	3320	2.111	1.887	45.3	FXPR	clay	10	hem	5			15	Weak	
1911-17	HW	56.4	4640	2.950	2.638	63.3	FXPR	clay	10	hem	5			15	Weak	
1911-18	HW	54	3740	2.378	2.126	51.0	FXPR	clay	10	hem	5			15	Weak	
1911-19	HW	51	1740	1.106	0.989	23.7	FXPR	clay	10	hem	5			15	Weak	
1911-20	HW	49	2500	1.589	1.421	34.1	BQFG	chl	20	hem	7			27	Weak	
1913-01	FW	76	4280	2.721	2.433	58.4	BQFG	hem	20	clay	25	lim	5	50	Moderate	
1913-02	FW	75.2	2660	1.691	1.512	36.3	BQFG	hem	20	clay	25	lim	5	50	Moderate	
1913-03	FW	74.4	3600	2.289	2.047	49.1	MIN	hem	20	clay	25	lim	5	50	Moderate	
1913-04	ORE	73.6	6400	4.069	3.638	87.3	MIN	hem	20	clay	25	lim	5	50	Moderate	
1913-05	ORE	73.2	4680	2.975	2.661	63.9	MIN	hem	20	clay	25	lim	5	50	Moderate	
1913-06	ORE	72.8	1360	0.865	0.773	18.6	MIN	hem	20	clay	25	lim	5	50	Moderate	
1913-07	ORE	70.4	2500	1.589	1.421	34.1	MIN	hem	20	clay	25	lim	5	50	Moderate	
1913-08	ORE	70	680	0.432	0.387	9.3	MIN	hem	20	clay	25	lim	5	50	Moderate	
1913-09	ORE	69.6	2000	1.272	1.137	27.3	MIN	hem	20	clay	25	lim	5	50	Moderate	
1913-10	ORE	67.2	3900	2.479	2.217	53.2	FXPR	chl	10	hem	1			11	Fresh	
1913-11	ORE	66.8	10740	6.828	6.106	146.5	FXPR	chl	10	hem	1			11	Fresh	
1913-12	ORE	66.4	12880	8.189	7.322	175.7	FXPR	chl	10	hem	1			11	Fresh	
1913-13	HW	66	12260	7.794	6.970	167.3	FXPR	chl	10	hem	1			11	Fresh	
1913-14	HW	65.6	21560	13.707	12.257	294.2	FXPR	chl	10	hem	1			11	Fresh	
1913-15	HW	65.2	17400	11.062	9.892	237.4	FXPR	chl	10	hem	1			11	Fresh	
1913-16	HW	64.8	13060	8.303	7.425	178.2	FXPR	chl	10	hem	1			11	Fresh	
1913-17	HW	64.4	23860	15.169	13.565	325.6	FXPR	chl	10	hem	1			11	Fresh	
1913-18	HW	60	12920	8.214	7.345	176.3	FXPR	chl	10	hem	1			11	Fresh	
1913-19	HW	57	9880	6.281	5.617	134.8	BQFG	chl	10	hem	1			11	Fresh	
1913-20	HW	54.5	14960	9.511	8.505	204.1	BQFG	chl	10	hem	1			11	Fresh	
1915-01	FW	80	4980	3.166	2.831	67.9	FXPR	ill	50	lim	3	hem	1	54	Moderate	
1915-02	FW	79.2	1720	1.094	0.978	23.5	MIN	lim	30	clay	10			40	Weak	

Test Number ("Hole"-"Test")	Segment Type	Position (m)	Test Result (kPa)	I_s	$I_{s\,50}$	UCS ($24*I_{s50}$)	Lithology	Min1	%	Min2	%	Min3	%	Total	Alteration	Comments
1915-04	ORE	77.6	1580	1.005	0.898	21.6	MIN	lim	30	clay	10			40	Weak	
1915-05	ORE	77.2	3300	2.098	1.876	45.0	MIN	lim	30	clay	10			40	Weak	
1915-06	ORE	76.8	4780	3.039	2.717	65.2	MIN	ill	50					50	Moderate	
1915-07	ORE	73.9	6160	3.916	3.502	84.0	MIN	ill	50	lim	2			52	Moderate	
1915-08	ORE	73.5	2920	1.856	1.660	39.8	MIN	ill	50	lim	2			52	Moderate	
1915-10	ORE	69.6	4160	2.645	2.365	56.8	HPGD	ill	25	hem	10			35	Weak	
1915-11	ORE	69.4	8840	5.620	5.026	120.6	HPGD	ill	25	hem	10			35	Weak	
1915-12	ORE	69	5220	3.319	2.968	71.2	HPGD	ill	25	hem	10			35	Weak	
1915-13	HW	68.6	6340	4.031	3.604	86.5	HPGD	ill	25	hem	10			35	Weak	
1915-14	HW	68.2	6100	3.878	3.468	83.2	HPGD	sil	30	ill	10			40	Weak	
1915-15	HW	67.8	8980	5.709	5.105	122.5	HPGD	sil	30	ill	10			40	Weak	
1915-16	HW	67.4	15120	9.613	8.596	206.3	HPGD	sil	30	ill	10			40	Weak	
1915-17	HW	67	10120	6.434	5.753	138.1	HPGD	sil	30	ill	10			40	Weak	
1915-18	HW	65	9360	5.951	5.321	127.7	FXPR	sil	30	ill	10			40	Weak	
1915-19	HW	63	8380	5.328	4.764	114.3	CALC	sil	30	ill	10			40	Weak	
1915-20	HW	61	3780	2.403	2.149	51.6	CALC							0	Fresh	
1917-01	FW	82.5	7780	4.946	4.423	106.2	BQFG	ill	25	hem	15			40	Weak	
1917-02	FW	81.7	7060	4.489	4.014	96.3	BQFG	ill	25	hem	15			40	Weak	
1917-03	FW	80.9	5060	3.217	2.877	69.0	BQFG	ill	25	hem	15			40	Weak	
1917-04	ORE	80.4	1240	0.788	0.705	16.9	MIN	lim	50	ill	15			65	Moderate	
1917-05	ORE	80	7800	4.959	4.434	106.4	MIN	lim	50	ill	15			65	Moderate	
1917-06	ORE	79.6	7740	4.921	4.400	105.6	MIN	lim	50	ill	15			65	Moderate	
1917-07	ORE	78.9	3280	2.085	1.865	44.8	MIN	lim	50	ill	15			65	Moderate	
1917-08	ORE	78.5	3780	2.403	2.149	51.6	MIN	lim	50	ill	15			65	Moderate	
1917-09	ORE	78.1	3060	1.945	1.740	41.8	MIN	lim	50	ill	15			65	Moderate	
1917-10	ORE	76.7	2560	1.628	1.455	34.9	MIN	lim	50	ill	15			65	Moderate	
1917-11	ORE	76.3	4200	2.670	2.388	57.3	BQFG	ill	25	hem	1			26	Weak	
1917-12	ORE	75.9	1960	1.246	1.114	26.7	BQFG	ill	25	hem	1			26	Weak	
1917-13	HW	75.4	4280	2.721	2.433	58.4	BQFG	ill	25	hem	1			26	Weak	
1917-14	HW	75	11160	7.095	6.345	152.3	BQFG	ill	25	hem	1			26	Weak	
1917-15	HW	74.6	10220	6.498	5.810	139.4	BQFG	ill	25	hem	1			26	Weak	
1917-16	HW	74.2	2740	1.742	1.558	37.4	BQFG	ill	25	hem	1			26	Weak	
1917-17	HW	73.8	6500	4.132	3.695	88.7	BQFG	ill	25	hem	1			26	Weak	
1917-18	HW	72.3	2560	1.628	1.455	34.9	BQFG	ill	25	hem	1			26	Weak	
1917-19	HW	69.5	7260	4.616	4.127	99.1	FXPR	chl	20					20	Weak	
1917-20	HW	66.8	5060	3.217	2.877	69.0	BQFG	chl	20					20	Weak	
1923-01	FW	89.9	1600	1.017	0.910	21.8	MIN	hem	15	lim	7	clay	7	29	Weak	
1923-02	FW	89.1	2200	1.399	1.251	30.0	MIN	hem	15	lim	7	clay	7	29	Weak	
1923-03	FW	88.3	2100	1.335	1.194	28.7	MIN	hem	15	lim	7	clay	7	29	Weak	
1923-04	ORE	87.9	3300	2.098	1.876	45.0	MIN	chl	10	lim	2	sil	15	27	Weak	
1923-05	ORE	87.5	1760	1.119	1.001	24.0	MIN	chl	10	lim	2	sil	15	27	Weak	
1923-06	ORE	87.1	400	0.254	0.227	5.5	MIN	chl	10	lim	2	sil	15	27	Weak	
1923-07	ORE	85.4	1520	0.966	0.864	20.7	MIN	hem	15	clay	15	chl	10	40	Weak	
1923-09	ORE	84.6	2140	1.361	1.217	29.2	MIN	hem	15	clay	15	chl	10	40	Weak	
1923-10	ORE	83.1	5460	3.471	3.104	74.5	FXPR	chl	15					15	Weak	
1923-11	ORE	82.7	4480	2.848	2.547	61.1	BQFG	chl	15					15	Weak	
1923-12	ORE	82.3	8620	5.480	4.901	117.6	BQFG	chl	15					15	Weak	
1923-13	HW	82	12180	7.744	6.924	166.2	BQFG	chl	15					15	Weak	
1923-14	HW	81.6	7180	4.565	4.082	98.0	BQFG	chl	15					15	Weak	
1923-15	HW	81.2	4240	2.696	2.410	57.9	BQFG	chl	15					15	Weak	
1923-17	HW	80.4	1960	1.246	1.114	26.7	BQFG	chl	15					15	Weak	
1923-18	HW	77.5	15740	10.007	8.948	214.8	FXPR	saus	7					7	Fresh	
1923-19	HW	75	7460	4.743	4.241	101.8	FXPR	saus	7					7	Fresh	
1923-20	HW	72.5	12500	7.947	7.106	170.6	FXPR	saus	7					7	Fresh	
1925-01	FW	87.9	1520	0.966	0.864	20.7	HPGD	sil	20	saus	20			40	Weak	
1925-02	FW	87.1	2460	1.564	1.399	33.6	HPGD	sil	20	saus	20			40	Weak	
1925-03	FW	86.3	6260	3.980	3.559	85.4	BQFG	sil	20	saus	20			40	Weak	
1925-07	ORE	84.7	4520	2.874	2.570	61.7	HPGD	saus	20	sil	20			40	Weak	
1925-08	ORE	84.3	5680	3.611	3.229	77.5	FLT	lim	15	sil	15			30	Weak	
1925-09	ORE	83.9	7480	4.756	4.252	102.1	FLT	lim	15	sil	15			30	Weak	
1925-13	HW	80.7	2480	1.577	1.410	33.8	FLT	chl	20	sil	15	clay	5	40	Weak	
1925-14	HW	80	3480	2.212	1.978	47.5	FXPR	chl	20	sil	15	clay	5	40	Weak	
1925-15	HW	79.7	5620	3.573	3.195	76.7	FXPR	chl	20	sil	15	clay	5	40	Weak	
1925-16	HW	79.5	8080	5.137	4.594	110.2	FXPR	chl	20	sil	15	clay	5	40	Weak	
1925-17	HW	79.1	3640	2.314	2.069	49.7	FXPR	chl	20	sil	15	clay	5	40	Weak	
1925-18	HW	77	5140	3.268	2.922	70.1	BQFG	chl	20	sil	15	clay	5	40	Weak	
1925-19	HW	75	2040	1.297	1.160	27.8	FLT	chl	20	sil	15	clay	7	42	Weak	
1925-20	HW	73	3720	2.365	2.115	50.8	BQFG	sil	10	chl	7			17	Weak	
1929-01	FW	77	10080	6.409	5.731	137.5	FXPR	sil	20	ep	7	chl	5	32	Weak	
1929-02	FW	76.2	5540	3.522	3.150	75.6	MIN	sil	20	ep	7	chl	5	32	Weak	
1929-03	FW	75.4	6480	4.120	3.684	88.4	QFGN	hem	20	chl	10	clay	5	35	Weak	
1929-04	ORE	74.9	5280	3.357	3.002	72.0	QFGN	hem	20	chl	10	clay	5	35	Weak	
1929-05	ORE	74.5	2240	1.424	1.273	30.6	QFGN	hem	20	chl	10	clay	5	35	Weak	
1929-06	ORE	74.1	7140	4.539	4.059	97.4	QFGN	hem	20	chl	10	clay	5	35	Weak	
1929-07	ORE	72.4	920	0.585	0.523	12.6	MIN	sil	20	lim	5			25	Weak	
1929-10	ORE	69.8	3320	2.111	1.887	45.3	MIN	chl	30	sil	20			50	Moderate	
1929-11	ORE	69.4	4660	2.963	2.649	63.6	MIN	hem	50	sil	25			75	Moderate	
1929-12	ORE	69	3680	2.340	2.092	50.2	MIN	hem	50	sil	25			75	Moderate	
1929-13	HW	68.7	2980	1.895	1.694	40.7	FXPR	chl	25					25	Weak	
1929-14	HW	68.3	5700	3.624	3.241	77.8	FXPR	chl	25					25	Weak	

Test Number ("Hole"-"Test")	Segment Type	Position (m)	Test Result (kPa)	I_s	$I_{s\,50}$	UCS ($24*I_{s50}$)	Lithology	Min1	%	Min2	%	Min3	%	Total	Alteration	Comments
1929-15	HW	67.9	8560	5.442	4.866	116.8	FXPR	chl	25					25	Weak	
1929-16	HW	67.5	7040	4.476	4.002	96.1	FXPR	chl	25					25	Weak	
1929-17	HW	67.1	8740	5.557	4.969	119.3	BQFG	chl	25					25	Weak	
1929-18	HW	65	13920	8.850	7.914	189.9	FXPR	chl	25					25	Weak	
1929-19	HW	63	9880	6.281	5.617	134.8	MIN	sil	30	hem	15	lim	5	50	Moderate	
1929-20	HW	61	11340	7.210	6.447	154.7	BQFG	sil	15	chl	15	py	1	31	Weak	
1931-01	FW	79	3060	1.945	1.740	41.8	BQFG	sil	15					15	Weak	
1931-02	FW	78.3	3500	2.225	1.990	47.8	MIN	sil	20	hem	10			30	Weak	
1931-03	FW	77.6	3760	2.390	2.138	51.3	FXPR	sil	20	hem	10			30	Weak	
1931-04	ORE	77.4	4140	2.632	2.354	56.5	FXPR	sil	20	hem	10			30	Weak	
1931-05	ORE	77	3080	1.958	1.751	42.0	MIN	sil	20	hem	10			30	Weak	
1931-06	ORE	76.6	3120	1.984	1.774	42.6	FXPR	sil	20	hem	10			30	Weak	
1931-07	ORE	73.4	2780	1.767	1.580	37.9	FXPR	clay	7	lim	5			12	Fresh	
1931-08	ORE	73	3200	2.034	1.819	43.7	FXPR	clay	7	lim	5			12	Fresh	
1931-09	ORE	72.6	2280	1.450	1.296	31.1	MIN	clay	7	lim	5			12	Fresh	
1931-10	ORE	69.2	2220	1.411	1.262	30.3	BQFG	chl	20	sil	10			30	Weak	
1931-11	ORE	68.8	4600	2.925	2.615	62.8	BQFG	chl	20	sil	10			30	Weak	
1931-12	ORE	68.4	3700	2.352	2.103	50.5	BQFG	chl	20	sil	10			30	Weak	
1931-13	HW	68	12080	7.680	6.868	164.8	FXPR	sil	15	chl	10			25	Weak	
1931-14	HW	67.6	13120	8.341	7.459	179.0	FXPR	sil	15	chl	10			25	Weak	
1931-15	HW	67.2	13360	8.494	7.595	182.3	FXPR	sil	15	chl	10			25	Weak	
1931-16	HW	66.8	15800	10.045	8.982	215.6	FXPR	sil	15	chl	10			25	Weak	
1931-17	HW	66.4	18240	11.596	10.370	248.9	FXPR	sil	15	chl	10			25	Weak	
1931-18	HW	64.5	8300	5.277	4.719	113.2	BQFG	sil	15	chl	10			25	Weak	
1931-19	HW	62.5	4920	3.128	2.797	67.1	FXPR	chl	7					7	Fresh	
1931-20	HW	60.5	11580	7.362	6.583	158.0	FXPR	chl	7					7	Fresh	
1933-01	FW	86	3840	2.441	2.183	52.4	BQFG	sil	15	saus	15			30	Weak	
1933-02	FW	85.2	9220	5.862	5.242	125.8	MIN	hem	65	lim	5	clay		70	Moderate	
1933-03	FW	84.4	3840	2.441	2.183	52.4	MIN	hem	55	lim	15	clay	5	75	Moderate	
1933-04	ORE	83.9	4260	2.708	2.422	58.1	MIN	hem	55	lim	15	clay	5	75	Moderate	
1933-05	ORE	83.5	1460	0.928	0.830	19.9	MIN	hem	55	lim	15	clay	5	75	Moderate	
1933-06	ORE	83.1	9720	6.180	5.526	132.6	MIN	hem	55	lim	15	clay	5	75	Moderate	
1933-07	ORE	78.4	3320	2.111	1.887	45.3	BQFG	sil	15	chl	5			20	Weak	
1933-08	ORE	78	4800	3.052	2.729	65.5	BQFG	sil	15	chl	5			20	Weak	
1933-09	ORE	77.6	940	0.598	0.534	12.8	MIN	sil	15	chl	5			20	Weak	
1933-10	ORE	73.4	4020	2.556	2.285	54.9	FXPR	sil	15	chl	7	hem	5	27	Weak	
1933-11	ORE	73	3140	1.996	1.785	42.8	FXPR	sil	15	chl	7	hem	5	27	Weak	
1933-12	ORE	72.6	6200	3.942	3.525	84.6	FXPR	sil	15	chl	7	hem	5	27	Weak	
1933-13	HW	72	4800	3.052	2.729	65.5	FXPR	chl	10	clay	5	hem	3	18	Weak	
1933-14	HW	71.6	4640	2.950	2.638	63.3	FXPR	chl	10	clay	5	hem	3	18	Weak	
1933-15	HW	71.2	3200	2.034	1.819	43.7	BQFG	sil	15	chl	10			25	Weak	
1933-16	HW	70.8	8500	5.404	4.832	116.0	BQFG	sil	15	chl	10			25	Weak	
1933-17	HW	70.4	15540	9.880	8.835	212.0	BQFG	sil	15	chl	10			25	Weak	
1933-18	HW	68	6700	4.260	3.809	91.4	BQFG	sil	15	chl	10			25	Weak	
1933-19	HW	66	15120	9.613	8.596	206.3	FXPR							0	Fresh	
1933-20	HW	64	15060	9.575	8.562	205.5	BQFG	chl	7					7	Fresh	
1935-01	FW	93	4640	2.950	2.638	63.3	FXPR	ill	40	lim	20	hem	7	67	Moderate	
1935-02	FW	92.2	3540	2.251	2.013	48.3	MIN	ill	40	lim	20	hem	7	67	Moderate	
1935-03	FW	91.4	3340	2.123	1.899	45.6	MIN	ill	40	lim	20	hem	7	67	Moderate	
1935-05	ORE	90.3	2780	1.767	1.580	37.9	MIN	ill	40	lim	20	hem	7	67	Moderate	
1935-06	ORE	89.9	1100	0.699	0.625	15.0	MIN	ill	40	lim	20	hem	7	67	Moderate	
1935-08	ORE	82.5	1820	1.157	1.035	24.8	MIN	ill	40	lim	20	hem	7	67	Moderate	
1935-09	ORE	82.1	3020	1.920	1.717	41.2	MIN	ill	40	lim	20	hem	7	67	Moderate	
1935-10	ORE	75.2	11180	7.108	6.356	152.5	FXPR	chl	10	sil	10			20	Weak	
1935-11	ORE	74.8	13060	8.303	7.425	178.2	FXPR	chl	10	sil	10			20	Weak	
1935-12	ORE	74.4	9160	5.824	5.208	125.0	FXPR	chl	10	sil	10			20	Weak	
1935-13	HW	74	12460	7.922	7.084	170.0	FXPR	chl	10	sil	10			20	Weak	
1935-14	HW	73.6	12420	7.896	7.061	169.5	FXPR	chl	10	sil	10			20	Weak	
1935-15	HW	73.2	10040	6.383	5.708	137.0	FXPR	chl	10	sil	10			20	Weak	
1935-16	HW	72.8	13740	8.735	7.811	187.5	FXPR	chl	10	sil	10			20	Weak	
1935-17	HW	72.4	15880	10.096	9.028	216.7	FXPR	chl	10	sil	10			20	Weak	
1935-18	HW	70	12660	8.049	7.197	172.7	FXPR							0	Fresh	
1935-19	HW	67.5	15460	9.829	8.789	210.9	CALC							0	Fresh	
1935-20	HW	64.5	17600	11.189	10.006	240.1	BQFG							0	Fresh	
1943-01	FW	90	2860	1.818	1.626	39.0	BQFG	ill	60	lim	7	hem	10	77	Moderate	
1943-02	FW	89.2	2980	1.895	1.694	40.7	MIN	ill	60	lim	7	hem	10	77	Moderate	
1943-03	FW	88.4	2100	1.335	1.194	28.7	FXPR	ill	60	lim	7	hem	10	77	Moderate	
1943-04	ORE	88	3920	2.492	2.229	53.5	FXPR	ill	60	lim	7	hem	10	77	Moderate	
1943-05	ORE	87.6	3540	2.251	2.013	48.3	FXPR	ill	60	lim	7	hem	10	77	Moderate	
1943-06	ORE	87.2	4460	2.836	2.536	60.9	FXPR	ill	60	lim	7	hem	10	77	Moderate	
1943-07	ORE	85.8	900	0.572	0.512	12.3	FXPR	ill	60	lim	7	hem	10	77	Moderate	
1943-09	ORE	85	1060	0.674	0.603	14.5	FXPR	ill	60	lim	7	hem	10	77	Moderate	
1943-10	ORE	83.5	2500	1.589	1.421	34.1	FXPR	ill	60	lim	7	hem	10	77	Moderate	
1943-11	ORE	83.1	5180	3.293	2.945	70.7	FXPR	ill	60	lim	7	hem	10	77	Moderate	
1943-12	ORE	82.7	7100	4.514	4.036	96.9	FXPR	ill	60	lim	7	hem	10	77	Moderate	
1943-13	HW	81.3	4860	3.090	2.763	66.3	FXPR	ill	60	lim	7	hem	10	77	Moderate	
1943-16	HW	80.1	5280	3.357	3.002	72.0	FXPR	ill	60	lim	7	hem	10	77	Moderate	
1943-17	HW	79.7	8540	5.429	4.855	116.5	FXPR	ill	20					20	Weak	
1943-18	HW	78	5660	3.598	3.218	77.2	FXPR	ill	20					20	Weak	
1943-19	HW	76	3040	1.933	1.728	41.5	FXPR	ill	20					20	Weak	

Test Number ("Hole"-"Test")	Segment Type	Position (m)	Test Result (kPa)	I_s	$I_{s\,50}$	UCS ($24{*}I_{s50}$)	Lithology	Min1	%	Min2	%	Min3	%	Total	Alteration	Comments
1943-20	HW	73.5	3180	2.022	1.808	43.4	FXPR	ill	20					20	Weak	
1949-01	FW	79	9480	6.027	5.389	129.3	FXPR	chl	15					15	Weak	
1949-02	FW	78.2	11280	7.171	6.413	153.9	FXPR	chl	15					15	Weak	
1949-03	FW	77.4	13600	8.646	7.732	185.6	FXPR	ill	20	sil	20	lim	7	47	Weak	
1949-10	ORE	74.3	6640	4.221	3.775	90.6	FXPR	ill	20	sil	20	lim	7	47	Weak	
1949-11	ORE	73.9	2480	1.577	1.410	33.8	FXPR	ill	20	sil	20	lim	7	47	Weak	
1949-12	ORE	73.5	1480	0.941	0.841	20.2	HPGD	ill	20	sil	20	lim	7	47	Weak	
1949-13	HW	73	3980	2.530	2.263	54.3	HPGD	ill	20	sil	20	lim	7	47	Weak	
1949-14	HW	72.6	4200	2.670	2.388	57.3	HPGD	ill	20	sil	20	lim	7	47	Weak	
1949-15	HW	72.2	6560	4.171	3.729	89.5	HPGD	ill	20	sil	20	lim	7	47	Weak	
1949-16	HW	71.8	16400	10.427	9.324	223.8	FXPR	chl	10	kspar	2			12	Fresh	
1949-17	HW	71.4	5840	3.713	3.320	79.7	FXPR	chl	10	kspar	2			12	Fresh	
1949-18	HW	69	11680	7.426	6.640	159.4	FXPR	chl	10	kspar	2			12	Fresh	
1949-19	HW	66	2480	1.577	1.410	33.8	BQFG	chl	10	kspar	2			12	Fresh	
1949-20	HW	64	9080	5.773	5.162	123.9	FXPR	chl	10	kspar	2			12	Fresh	
1951-01	FW	84.6	17120	10.884	9.733	233.6	BQFG	sil	50	chl	15			65	Moderate	
1951-02	FW	83.8	17880	11.367	10.165	244.0	BQFG	sil	50	chl	15			65	Moderate	
1951-03	FW	83	6000	3.815	3.411	81.9	FXPR	hem	30	hem	20	ill	7	57	Moderate	
1951-07	ORE	79.8	11240	7.146	6.390	153.4	FXPR	chl	10	hem	10			20	Weak	
1951-08	ORE	79.2	14720	9.358	8.369	200.8	FXPR	chl	10	hem	10			20	Weak	
1951-09	ORE	77.4	8080	5.137	4.594	110.2	FXPR	chl	5					5	Fresh	
1951-13	HW	73.8	13900	8.837	7.902	189.7	FXPR	chl	3					3	Fresh	
1951-14	HW	73.4	15040	9.562	8.550	205.2	FXPR	chl	3					3	Fresh	
1951-15	HW	73	14720	9.358	8.369	200.8	FXPR	chl	3					3	Fresh	
1951-16	HW	72.6	17860	11.355	10.154	243.7	FXPR	chl	3					3	Fresh	
1951-17	HW	72.2	17620	11.202	10.017	240.4	FXPR	chl	3					3	Fresh	
1951-18	HW	71	13320	8.468	7.573	181.7	FXPR	chl	3					3	Fresh	
1951-19	HW	69	20480	13.020	11.643	279.4	FXPR	chl	3					3	Fresh	
1951-20	HW	68	11640	7.400	6.617	158.8	FXPR	chl	3					3	Fresh	
1953-01	FW	96	3980	2.530	2.263	54.3	BQFG	ill	40	lim	3			43	Weak	
1953-02	FW	95.2	3840	2.441	2.183	52.4	BQFG	ill	40	lim	3			43	Weak	
1953-03	FW	94.4	7040	4.476	4.002	96.1	BQFG	ill	40	lim	3			43	Weak	
1953-04	ORE	94	4120	2.619	2.342	56.2	FXPR	ill	40	lim	3			43	Weak	
1953-05	ORE	93.6	3980	2.530	2.263	54.3	FXPR	ill	40	lim	3			43	Weak	
1953-06	ORE	93.2	3100	1.971	1.762	42.3	MIN	lim	75					75	Moderate	
1953-09	ORE	90.6	3700	2.352	2.103	50.5	FXPR	ill	50	sil	10	clay	3	63	Moderate	
1953-10	ORE	88.8	6340	4.031	3.604	86.5	FXPR	ill	50	sil	10	clay	3	63	Moderate	
1953-11	ORE	88.4	4000	2.543	2.274	54.6	FXPR	ill	50	sil	10	clay	3	63	Moderate	
1953-12	ORE	88	7380	4.692	4.196	100.7	FXPR	ill	50	sil	10	clay	3	63	Moderate	
1953-13	HW	87.5	9960	6.332	5.662	135.9	FXPR	ill	50	sil	10	clay	3	63	Moderate	
1953-14	HW	87.1	4280	2.721	2.433	58.4	FXPR	ill	50	sil	10	clay	3	63	Moderate	
1953-15	HW	86.7	7020	4.463	3.991	95.8	FXPR	ill	50	sil	10	clay	3	63	Moderate	
1953-16	HW	86.3	6040	3.840	3.434	82.4	FXPR	ill	50	sil	10	clay	3	63	Moderate	
1953-17	HW	85.9	10080	6.409	5.731	137.5	FXPR	ill	50	sil	10	clay	3	63	Moderate	
1953-18	HW	84	3580	2.276	2.035	48.8	FXPR	ill	50	sil	10	clay	3	63	Moderate	
1953-19	HW	81.5	11360	7.222	6.458	155.0	FXPR	chl	20					20	Weak	
1953-20	HW	79	9500	6.040	5.401	129.6	FXPR	chl	20					20	Weak	
2276 Tested May 6-13, 2005																
2276-01	HW	84.9	12300	7.820	6.993	167.8	BQFG	clay	10	sil	7	chl	5	22	Weak	
2276-02	HW	86.2	11920	7.578	6.777	162.6	FXPR	clay	10	sil	7	chl	5	22	Weak	
2276-03	HW	86.7	6560	4.171	3.729	89.5	FXPR	clay	10	sil	7	chl	5	22	Weak	
2276-04	ORE	87	6040	3.840	3.434	82.4	FXPR	clay	10	sil	7	chl	5	22	Weak	
2276-05	ORE	87.7	2880	1.831	1.637	39.3	FXPR	clay	40	ser	10	hem	4	54	Moderate	
2276-06	ORE	88	2980	1.895	1.694	40.7	FXPR	clay	40	ser	10	hem	4	54	Moderate	
2276-07	ORE	88.7	1320	0.839	0.750	18.0	FXPR	clay	40	ser	10	hem	4	54	Moderate	
2276-08	ORE	88.9	900	0.572	0.512	12.3	FXPR	clay	40	ser	10	hem	4	54	Moderate	
2276-09	ORE	89.1	2300	1.462	1.308	31.4	FXPR	clay	40	ser	10	hem	4	54	Moderate	
2276-10	FW	89.7	5000	3.179	2.843	68.2	FXPR	clay	40	ser	10	hem	4	54	Moderate	
2276-11	FW	90.3	120	0.076	0.068	1.6	FXPR	clay	40	ser	10	hem	4	54	Moderate	
2276-12	FW	90.8	3040	1.933	1.728	41.5	FXPR	clay	40	ser	10	hem	4	54	Moderate	
2276-13	FW	91.4	600	0.381	0.341	8.2	FXPR	clay	40	ser	10	hem	4	54	Moderate	
2276-14	FW	92.3	1960	1.246	1.114	26.7	BQFG	clay	20	sil	12	ser	7	39	Weak	
2276-15	FW	93	7300	4.641	4.150	99.6	BQFG	clay	20	sil	12	ser	7	39	Weak	
2276-16	FW	96.3	18980	12.067	10.790	259.0	BQFG	sil	40	clay	10			50	Moderate	
2276-17	FW	97.7	9700	6.167	5.515	132.3	BQFG	clay	20	sil	20	ser	5	45	Weak	
2276-18	FW	99.1	10240	6.510	5.822	139.7	BQFG	clay	20	sil	20	ser	5	45	Weak	
2277 Tested May 6-13, 2005																
2277-01	HW	68	8560	5.442	4.866	116.8	AGRP	clay	25	chl	15	ser	10	50	Moderate	
2277-02	HW	68.6	11960	7.604	6.799	163.2	AGRP	clay	25	chl	15	ser	10	50	Moderate	
2277-03	HW	69.5	3600	2.289	2.047	49.1	AGRP	clay	25	chl	15	ser	10	50	Moderate	
2277-04	ORE	70.2	3300	2.098	1.876	45.0	AGRP	clay	25	chl	15	ser	10	50	Moderate	
2277-05	ORE	70.3	200	0.127	0.114	2.7	AGRP	clay	25	chl	15	ser	10	50	Moderate	
2277-06	ORE	70.5	2860	1.818	1.626	39.0	AGRP	hem	65	lim	15	clay	5	85	Moderate	
2277-07	ORE	70.6	2680	1.704	1.524	36.6	AGRP	hem	65	lim	15	clay	5	85	Moderate	
2277-08	ORE	71	7320	4.654	4.162	99.9	AGRP	hem	65	lim	15	clay	5	85	Moderate	
2277-09	FW/HW	71.4	2340	1.488	1.330	31.9	FXPR	clay	30	chl	15	ser	10	55	Moderate	
2277-10	FW/HW	71.9	2920	1.856	1.660	39.8	FXPR	clay	30	chl	15	ser	10	55	Moderate	
2277-11	FW/HW	72.2	9400	5.976	5.344	128.3	FXPR	clay	30	chl	15	ser	10	55	Moderate	
2277-12	FW/HW	73.2	1900	1.208	1.080	25.9	FXPR	clay	30	chl	15	ser	10	55	Moderate	
2277-13	FW/HW	73.7	2420	1.539	1.376	33.0	FXPR	clay	30	chl	15	ser	10	55	Moderate	
2277-14	FW/HW	74.4	8640	5.493	4.912	117.9	FXPR	clay	30	chl	15	ser	10	55	Moderate	

Test Number ("Hole"-"Test")	Segment Type	Position (m)	Test Result (kPa)	I_s	$I_{s\,50}$	UCS ($24*I_{s50}$)	Lithology	Min1	%	Min2	%	Min3	%	Total	Alteration	Comments
2277-15	FW/HW	75.7	5240	3.331	2.979	71.5	AGRP	chl	10	clay	5	hem	2	17	Weak	
2277-16	FW/HW	76.3	3300	2.098	1.876	45.0	AGRP	chl	10	clay	5	hem	2	17	Weak	
2277-17	FW/HW	77.4	4240	2.696	2.410	57.9	AGRP	chl	10	clay	5	hem	2	17	Weak	
2277-18	FW/HW	77.6	3700	2.352	2.103	50.5	FXPR	sil	20	chl	12	clay	7	39	Weak	
2277-19	ORE	78.3	10780	6.854	6.129	147.1	FXPR	sil	20	chl	12	clay	7	39	Weak	
2277-20	ORE	78.4	9960	6.332	5.662	135.9	FXPR	sil	20	chl	12	clay	7	39	Weak	
2277-21	ORE	78.7	900	0.572	0.512	12.3	FXPR	sil	20	chl	12	clay	7	39	Weak	
2277-22	ORE	79.3	2720	1.729	1.546	37.1	FXPR	sil	20	chl	12	clay	7	39	Weak	
2277-23	FW/HW	80.6	12260	7.794	6.970	167.3	FXPR	sil	20	chl	12	clay	7	39	Weak	
2277-24	FW/HW	82	18760	11.927	10.665	256.0	FXPR	sil	20	chl	12	clay	7	39	Weak	
2277-25	FW/HW	83.9	16760	10.655	9.528	228.7	FXPR	sil	20	chl	12	clay	7	39	Weak	
2277-26	FW/HW	84.5	9440	6.002	5.367	128.8	FXPR	sil	20	chl	12	clay	7	39	Weak	
2277-27	FW/HW	85.4	2920	1.856	1.660	39.8	FXPR	sil	20	chl	12	clay	7	39	Weak	
2277-28	FW/HW	87.3	12500	7.947	7.106	170.6	AGRP	hem	30	chl	15	lim	15	60	Moderate	+10% clay
2277-29	FW/HW	87.6	1780	1.132	1.012	24.3	AGRP	hem	30	chl	15	lim	15	60	Moderate	+10% clay
2277-30	FW/HW	89.5	4220	2.683	2.399	57.6	AGRP	hem	30	chl	15	lim	15	60	Moderate	+10% clay
2277-31	FW/HW	90	5760	3.662	3.275	78.6	AGRP	hem	30	chl	15	lim	15	60	Moderate	+10% clay
2277-32	FW/HW	90.3	4140	2.632	2.354	56.5	AGRP	hem	30	chl	15	lim	15	60	Moderate	+10% clay
2277-33	FW/HW	90.9	15760	10.020	8.960	215.0	AGRP	hem	30	chl	15	lim	15	60	Moderate	+10% clay
2277-34	ORE	91.7	6660	4.234	3.786	90.9	AGRP	hem	30	chl	15	lim	15	60	Moderate	+10% clay
2277-35	ORE	92	1620	1.030	0.921	22.1	AGRP	hem	30	chl	15	lim	15	60	Moderate	+10% clay
2277-36	ORE	92.8	3780	2.403	2.149	51.6	AGRP	hem	30	lim	15	clay	20	65	Moderate	+12% chl
2277-37	ORE	93.6	5540	3.522	3.150	75.6	AGRP	hem	30	lim	15	clay	20	65	Moderate	+12% chl
2277-38	ORE	94.9	6000	3.815	3.411	81.9	AGRP	hem	30	lim	15	clay	20	65	Moderate	+12% chl
2277-39	ORE	95.9	4820	3.064	2.740	65.8	AGRP	hem	30	lim	15	clay	20	65	Moderate	+12% chl
2277-40	ORE	97.8	6780	4.310	3.855	92.5	AGRP	hem	30	lim	15	clay	20	65	Moderate	+12% chl
2277-41	FW	98.8	2080	1.322	1.183	28.4	AGRP	hem	30	lim	15	clay	20	65	Moderate	+12% chl
2277-42	FW	100.8	2500	1.589	1.421	34.1	FXPR	clay	25	chl	15	sil	10	50	Moderate	+10% ser / 7% hem
2277-43	FW	101.9	2520	1.602	1.433	34.4	FXPR	clay	25	chl	15	sil	10	50	Moderate	+10% ser / 7% hem
2277-44	FW	102.8	9420	5.989	5.355	128.5	FXPR	clay	25	chl	15	sil	10	50	Moderate	+10% ser / 7% hem
2277-45	FW	103.1	7820	4.972	4.446	106.7	FXPR	clay	25	chl	15	sil	10	50	Moderate	+10% ser / 7% hem
2277-46	FW	104.5	7980	5.073	4.537	108.9	FXPR	clay	15	chl	10	sil	7	32	Weak	
2277-47	FW	104.9	5920	3.764	3.366	80.8	FXPR	clay	15	chl	10	sil	7	32	Weak	
2277-48	FW	106	7060	4.489	4.014	96.3	FXPR	clay	15	chl	10	sil	7	32	Weak	
2277-49	FW	106.5	5840	3.713	3.320	79.7	AGRP	hem	15	lim	12	clay	10	37	Weak	
2277-50	FW	107.7	180	0.114	0.102	2.5	AGRP	hem	15	lim	12	clay	10	37	Weak	
2278 Tested May 17, 2005																
2278-01	HW	115.8	9200	5.849	5.230	125.5	AGRP	chl	17	clay	7	hem	3	27	Weak	
2278-02	HW	117.1	18100	11.507	10.290	247.0	AGRP	chl	17	clay	7	hem	3	27	Weak	
2278-03	HW	118.2	16820	10.694	9.562	229.5	AGRP	chl	17	clay	7	hem	3	27	Weak	
2278-04	HW	120.1	16720	10.630	9.506	228.1	AGRP	chl	17	clay	7	hem	3	27	Weak	
2278-05	HW	121.5	13740	8.735	7.811	187.5	AGRP	chl	17	clay	7	hem	3	27	Weak	
2278-06	HW	123.1	13400	8.519	7.618	182.8	AGRP	clay	50	hem	25	chl	10	85	Moderate	ffl
2278-07	HW	123.8	3920	2.492	2.229	53.5	AGRP	clay	50	hem	25	chl	10	85	Moderate	ffl
2278-08	HW	124.4	3560	2.263	2.024	48.6	AGRP	clay	50	hem	25	chl	10	85	Moderate	ffl
2278-09	HW	124.9	4160	2.645	2.365	56.8	AGRP	clay	50	hem	25	chl	10	85	Moderate	ffl
2278-10	HW	125.6	2780	1.767	1.580	37.9	AGRP	clay	50	hem	25	chl	10	85	Moderate	ffl
2278-11	ORE	125.9	5340	3.395	3.036	72.9	AGRP	clay	50	hem	25	chl	10	85	Moderate	ffl
2278-12	ORE	126.2	4560	2.899	2.592	62.2	AGRP	clay	50	hem	25	chl	10	85	Moderate	ffl
2278-13	ORE	126.8	780	0.496	0.443	10.6	AGRP	clay	50	hem	25	chl	10	85	Moderate	ffl
2278-14	ORE	127.2	1640	1.043	0.932	22.4	AGRP	hem	40	clay	30	lim	15	85	Moderate	ffl
2278-15	ORE	127.6	3060	1.945	1.740	41.8	FXPR	hem	40	clay	30	lim	15	85	Moderate	ffl
2278-16	ORE	128.1	4100	2.607	2.331	55.9	FXPR	hem	40	clay	30	lim	15	85	Moderate	ffl
2278-17	ORE	128.6	3580	2.276	2.035	48.8	FXPR	hem	40	clay	30	lim	15	85	Moderate	ffl
2278-18	FW	129.3	8440	5.366	4.798	115.2	FXPR	chl	25	ser	15	sil	10	50	Moderate	
2278-19	FW	130.1	3720	2.365	2.115	50.8	FXPR	chl	25	ser	15	sil	10	50	Moderate	
2278-20	FW	131.4	9960	6.332	5.662	135.9	FXPR	chl	25	ser	15	sil	10	50	Moderate	
2278-21	FW	133	5580	3.548	3.172	76.1	FXPR	chl	25	ser	15	sil	10	50	Moderate	
2279 Tested May 18, 2005																
2279-01	HW	85.5	6120	3.891	3.479	83.5	FXPR	chl	17	clay	12	py	3	32	Weak	
2279-02	HW	86.8	13540	8.608	7.698	184.7	FXPR	chl	17	clay	12	py	3	32	Weak	
2279-03	HW	88.4	16040	10.198	9.119	218.9	AGRP	chl	20	clay	10	hem	3	33	Weak	
2279-04	HW	89.9	14080	8.952	8.005	192.1	AGRP	chl	20	clay	10	hem	3	33	Weak	
2279-05	HW	91.2	16900	10.744	9.608	230.6	AGRP	chl	20	clay	10	hem	3	33	Weak	
2279-06	HW	92.6	10880	6.917	6.185	148.4	AGRP	chl	20	clay	10	hem	3	33	Weak	
2279-07	HW	93.2	5180	3.293	2.945	70.7	AGRP	chl	20	clay	10	hem	3	33	Weak	
2279-08	HW	93.8	11620	7.388	6.606	158.5	AGRP	chl	20	clay	10	hem	3	33	Weak	
2279-09	HW	94.4	7400	4.705	4.207	101.0	AGRP	chl	20	clay	10	hem	3	33	Weak	
2279-10	HW	95	2420	1.539	1.376	33.0	AGRP	hem	50	clay	20	lim	15	85	Moderate	ffl
2279-11	ORE	95.5	900	0.572	0.512	12.3	AGRP	hem	50	clay	20	lim	15	85	Moderate	ffl
2279-12	ORE	96.2	2880	1.831	1.637	39.3	AGRP	hem	50	clay	20	lim	15	85	Moderate	ffl
2279-13	ORE	96.4	5380	3.420	3.059	73.4	AGRP	hem	50	clay	20	lim	15	85	Moderate	ffl
2279-14	FW	96.6	5080	3.230	2.888	69.3	AGRP	hem	50	clay	20	lim	15	85	Moderate	ffl
2279-15	FW	97.6	9040	5.747	5.139	123.3	AGRP	chl	17	clay	12			29	Weak	
2279-16	FW	98.7	11200	7.121	6.367	152.8	AGRP	chl	17	clay	12			29	Weak	
2279-17	FW	99.7	5980	3.802	3.400	81.6	AGRP	chl	17	clay	12			29	Weak	
2279-18	FW	100.7	7780	4.946	4.423	106.2	AGRP	chl	17	clay	12			29	Weak	
2253 Tested May 22, 2005																
2253-01	HW	42.2	160	0.102	0.091	2.2	BQFG	clay	35	sil	15			50	Moderate	
2253-02	HW	43.5	14680	9.333	8.346	200.3	FXPR	chl	10	clay	5	py	2	17	Weak	
2253-03	HW	44.9	6960	4.425	3.957	95.0	HPGD	sil	20	chl	8	ser	7	35	Weak	
2253-04	HW	46.3	6260	3.980	3.559	85.4	HPGD	sil	20	chl	8	ser	7	35	Weak	
2253-05	HW	47.8	7640	4.857	4.343	104.2	FXPR	sil	20	chl	8	ser	7	35	Weak	
2253-06	HW	49.2	1100	0.699	0.625	15.0	FXPR	sil	20	chl	8	ser	7	35	Weak	

Test Number ("Hole"-"Test")	Segment Type	Position (m)	Test Result (kPa)	I_s	$I_{s\,50}$	UCS ($24*I_{s50}$)	Lithology	Min1	%	Min2	%	Min3	%	Total	Alteration	Comments
2253-07	HW	50.8	6140	3.904	3.491	83.8	QFGN	clay	30	ser	10	hem	2	42	Weak	
2253-08	ORE	51.8	2680	1.704	1.524	36.6	QFGN	clay	30	ser	10	hem	2	42	Weak	
2253-09	ORE	52.3	2100	1.335	1.194	28.7	FXPR	clay	30	ser	10	hem	2	42	Weak	
2253-10	ORE	52.9	420	0.267	0.239	5.7	FXPR	clay	50	hem	5			55	Moderate	
2253-11	ORE	53.6	1000	0.636	0.569	13.6	FXPR	clay	35	hem	2			37	Weak	
2253-12	ORE	54.7	1060	0.674	0.603	14.5	FXPR	clay	35	hem	2			37	Weak	
2253-13	ORE	55.1	2280	1.450	1.296	31.1	FXPR	clay	35	hem	2			37	Weak	
2253-14	ORE	55.4	2040	1.297	1.160	27.8	FXPR	clay	35	hem	2			37	Weak	
2253-15	FW	56.1	900	0.572	0.512	12.3	FXPR	clay	35	hem	2			37	Weak	
2253-16	FW	56.9	5460	3.471	3.104	74.5	FXPR	clay	35	hem	2			37	Weak	
2253-17	FW	57.5	4320	2.747	2.456	58.9	FXPR	clay	35	hem	2			37	Weak	
2253-18	FW	58.3	4280	2.721	2.433	58.4	FXPR	clay	35	hem	2			37	Weak	
2264 Tested May 22, 2005																
2264-01	HW	76.5	8260	5.251	4.696	112.7	AGRP	chl	12	sil	7	hem	5	24	Weak	
2264-02	HW	77.4	11260	7.159	6.401	153.6	AGRP	chl	12	sil	7	hem	5	24	Weak	
2264-03	HW	79.2	16820	10.694	9.562	229.5	AGRP	chl	12	sil	7	hem	5	24	Weak	
2264-04	HW	80.3	2800	1.780	1.592	38.2	AGRP	chl	12	sil	7	hem	5	24	Weak	
2264-05	HW	82.1	5920	3.764	3.366	80.8	AGRP	chl	12	sil	7	hem	5	24	Weak	
2264-06	HW	83.1	11960	7.604	6.799	163.2	AGRP	chl	12	sil	7	hem	5	24	Weak	
2264-07	HW	84.6	17420	11.075	9.903	237.7	AGRP	chl	12	sil	7	hem	5	24	Weak	
2264-08	HW	85.7	10640	6.765	6.049	145.2	AGRP	chl	12	sil	7	hem	5	24	Weak	
2264-09	ORE	86.4	7520	4.781	4.275	102.6	AGRP	chl	12	sil	7	hem	5	24	Weak	
2264-10	ORE	86.9	140	0.089	0.080	1.9	AGRP	chl	12	sil	7	hem	5	24	Weak	
2264-11	ORE	86.8	6040	3.840	3.434	82.4	AGRP	chl	12	sil	7	hem	5	24	Weak	
2264-12	ORE	87.1	3040	1.933	1.728	41.5	AGRP	chl	12	sil	7	hem	5	24	Weak	
2264-13	FW/HW	87.4	1860	1.183	1.057	25.4	AGRP	chl	12	sil	7	hem	5	24	Weak	
2264-14	FW/HW	89.3	17760	11.291	10.097	242.3	AGRP	chl	12	sil	7	hem	5	24	Weak	
2264-15	FW/HW	90.3	6500	4.132	3.695	88.7	AGRP	chl	12	sil	7	hem	5	24	Weak	
2264-16	FW/HW	91.4	16980	10.795	9.653	231.7	AGRP	chl	12	sil	7	hem	5	24	Weak	
2264-17	FW/HW	92.3	14960	9.511	8.505	204.1	AGRP	chl	12	sil	7	hem	5	24	Weak	
2264-18	ORE	93.7	2800	1.780	1.592	38.2	FXPR	chl	15	clay	10	py	5	30	Weak	
2264-19	ORE	94.4	6840	4.349	3.889	93.3	FXPR	clay	45	chl	15	hem	10	70	Moderate	+10% lim/ 5% si /graphitic/bleaching /mineralized flt zone
2264-20	ORE	94.5	8400	5.340	4.776	114.6	FXPR	clay	45	chl	15	hem	10	70	Moderate	+10% lim/ 5% si /graphitic/bleaching /mineralized flt zone
2264-21	ORE	94.7	1900	1.208	1.080	25.9	FXPR	clay	45	chl	15	hem	10	70	Moderate	+10% lim/ 5% si /graphitic/bleaching /mineralized flt zone
2264-22	ORE	95	1820	1.157	1.035	24.8	FXPR	clay	45	chl	15	hem	10	70	Moderate	+10% lim/ 5% si /graphitic/bleaching /mineralized flt zone
2264-23	FW	95.3	740	0.470	0.421	10.1	FXPR	clay	45	chl	15	hem	10	70	Moderate	+10% lim/ 5% si /graphitic/bleaching /mineralized flt zone
2264-24	FW	95.7	1780	1.132	1.012	24.3	FXPR	clay	45	chl	15	hem	10	70	Moderate	+10% lim/ 5% si /graphitic/bleaching /mineralized flt zone
2264-25	FW	96.3	5860	3.726	3.331	80.0	FXPR	clay	45	chl	15	hem	10	70	Moderate	+10% lim/ 5% si /graphitic/bleaching /mineralized flt zone
2249 Tested May 26 and June 7, 2005																
2249-01	HW	46.6	5120	3.255	2.911	69.9	FXPR	chl	10	hem	1	lim	1	12	Fresh	
2249-02	HW	47.5	9620	6.116	5.469	131.3	FXPR	chl	10	hem	1	lim	1	12	Fresh	
2249-03	HW	48.7	10860	6.904	6.174	148.2	FXPR	chl	10	hem	1	lim	1	12	Fresh	
2249-04	HW	50.6	13800	8.774	7.845	188.3	FXPR	chl	10	hem	1	lim	1	12	Fresh	
2249-05	HW	51.4	2600	1.653	1.478	35.5	FXPR	ill	30	hem	20	lim	10	60	Moderate	
2249-06	HW	53.2	860	0.547	0.489	11.7	FXPR	ill	30	hem	20	lim	10	60	Moderate	
2249-07	HW	53.8	8380	5.328	4.764	114.3	FXPR	ill	30	hem	20	lim	10	60	Moderate	
2249-08	HW	54.4	460	0.292	0.262	6.3	FXPR	ill	30	hem	20	lim	10	60	Moderate	
2249-09	HW	55	2490	1.583	1.416	34.0	FXPR	ill	30	hem	20	lim	10	60	Moderate	
2249-10	HW	56	1360	0.865	0.773	18.6	FXPR	ill	30	hem	20	lim	10	60	Moderate	
2249-11	ORE	56.4	960	0.610	0.546	13.1	FXPR	ill	30	hem	20	lim	10	60	Moderate	
2249-13	ORE	57.5	340	0.216	0.193	4.6	FXPR	ill	30	hem	20	lim	10	60	Moderate	
2249-14	ORE	57.9	420	0.267	0.239	5.7	FXPR	ill	30	hem	20	lim	10	60	Moderate	
2249-15	FW	59.1	1960	1.246	1.114	26.7	FXPR	ill	30	hem	20	lim	10	60	Moderate	
2249-16	FW	60.4	5600	3.560	3.184	76.4	HPGD	ill	30	hem	20	lim	10	60	Moderate	
2249-17	FW	61	3260	2.073	1.853	44.5	HPGD	ill	30	hem	20	lim	10	60	Moderate	
2288 tested June 7, 2005																
2288-01	HW	76.3	10740	6.828	6.106	146.5	FXPR	clay	12	ser	5			17	Weak	
2288-02	HW	77	9980	6.345	5.674	136.2	FXPR	clay	12	ser	5			17	Weak	
2288-03	HW	77.9	7400	4.705	4.207	101.0	FXPR	clay	12	ser	5			17	Weak	
2288-04	HW	79.9	10540	6.701	5.992	143.8	HPGD	clay	12	ser	5			17	Weak	
2288-05	HW	82	960	0.610	0.546	13.1	BQFG	clay	12	ser	5			17	Weak	
2288-06	HW	83.3	5240	3.331	2.979	71.5	BQFG	clay	12	ser	5			17	Weak	
2288-07	HW	84.2	2260	1.437	1.285	30.8	BQFG	clay	15	ser	10	hem	1	26	Weak	
2288-08	HW	85.3	5620	3.573	3.195	76.7	BQFG	clay	15	ser	10	hem	1	26	Weak	
2288-09	HW	85.7	5760	3.662	3.275	78.6	FXPR	clay	15	ser	10	hem	1	26	Weak	
2288-10	HW	86.2	3020	1.920	1.717	41.2	BQFG	clay	15	ser	10	hem	1	26	Weak	
2288-12	ORE	87.1	1780	1.132	1.012	24.3	BQFG	clay	40	hem	5	ser	5	50	Moderate	
2288-14	ORE	88.7	1560	0.992	0.887	21.3	BQFG	clay	40	hem	5	ser	5	50	Moderate	
2288-15	ORE	90.3	7120	4.527	4.048	97.1	HPGD	clay	35	ser	10			45	Weak	
2288-16	ORE	91	1020	0.648	0.580	13.9	BQFG	clay	35	ser	10			45	Weak	
2288-17	ORE	92.9	4780	3.039	2.717	65.2	BQFG	clay	35	ser	10			45	Weak	
2288-18	ORE	94.6	8240	5.239	4.685	112.4	BQFG	clay	25	ser	10	hem	3	38	Weak	
2288-19	ORE	95.5	20	0.013	0.011	0.3	BQFG	clay	25	ser	10	hem	3	38	Weak	
2288-20	ORE	96.6	1440	0.916	0.819	19.6	BQFG	clay	25	ser	10	hem	3	38	Weak	
2288-21	ORE	97.1	6000	3.815	3.411	81.9	BQFG	clay	12	ser	10	hem	2	24	Weak	
2288-22	ORE	98.3	4980	3.166	2.831	67.9	BQFG	clay	12	ser	10	hem	2	24	Weak	

Test Number ("Hole"-"Test")	Segment Type	Position (m)	Test Result (kPa)	I_s	$I_{s\,50}$	UCS (24*I_{s50})	Lithology	Min1	%	Min2	%	Min3	%	Total	Alteration	Comments
2288-23	FW	98.9	6020	3.827	3.422	82.1	BQFG	clay	12	ser	10	hem	2	24	Weak	
2288-24	FW	99.5	4180	2.658	2.376	57.0	BQFG	clay	12	ser	10	hem	2	24	Weak	
2288-25	FW	99.9	5980	3.802	3.400	81.6	BQFG	clay	12	ser	10	hem	2	24	Weak	
2288-26	FW	100.2	7220	4.590	4.105	98.5	BQFG	clay	12	ser	10	hem	2	24	Weak	
2245 tested June 7 and 10, 2005																
2245-01	HW	154.2	7300	4.641	4.150	99.6	HPGD	clay	25	chl	10	ser	10	45	Weak	
2245-02	HW	155.5	7580	4.819	4.309	103.4	HPGD	clay	25	chl	10	ser	10	45	Weak	
2245-03	HW	157.3	3460	2.200	1.967	47.2	BQFG	clay	25	chl	10	ser	10	45	Weak	
2245-04	HW	158.5	1920	1.221	1.092	26.2	BQFG	clay	25	chl	10	ser	10	45	Weak	
2245-05	HW	160.9	940	0.598	0.534	12.8	HPGD	clay	40	chl	2	hem	1	43	Weak	
2245-06	HW	161.5	1680	1.068	0.955	22.9	HPGD	clay	40	chl	2	hem	1	43	Weak	
2245-07	HW	161.9	1800	1.144	1.023	24.6	HPGD	clay	40	chl	2	hem	1	43	Weak	
2245-08	HW	162.6	500	0.318	0.284	6.8	HPGD	clay	40	chl	2	hem	1	43	Weak	
2245-09	HW	163.6	2380	1.513	1.353	32.5	HPGD	clay	40	chl	2	hem	1	43	Weak	
2245-10	ORE	164.9	1320	0.839	0.750	18.0	HPGD	clay	40	chl	2	hem	1	43	Weak	
2245-11	ORE	165.6	1860	1.183	1.057	25.4	HPGD	clay	40	chl	2	hem	1	43	Weak	
2245-12	ORE	166.2	1140	0.725	0.648	15.6	HPGD	clay	40	chl	2	hem	1	43	Weak	
2245-13	ORE	166.8	280	0.178	0.159	3.8	HPGD	clay	40	chl	2	hem	1	43	Weak	
2245-15	ORE	168.3	600	0.381	0.341	8.2	BQFG	clay	70					70	Moderate	
2245-17	FW	169	2320	1.475	1.319	31.7	BQFG	clay	45	chl	5	ser	5	55	Moderate	
2245-18	FW	170	3880	2.467	2.206	52.9	BQFG	clay	45	chl	5	ser	5	55	Moderate	
2245-19	FW	170.4	1700	1.081	0.966	23.2	BQFG	clay	45	chl	5	ser	5	55	Moderate	
2245-20	FW	170.8	1020	0.648	0.580	13.9	BQFG	clay	45	chl	5	ser	5	55	Moderate	
2299 tested June 10, 2005																
2299-01	FAR HW	103.1	1180	0.750	0.671	16.1	AGRP	chl	12	sil	7	py	5	24	Weak	
2299-02	FAR HW	104.5	13480	8.570	7.664	183.9	HPGD	sil	20	saus	15	chl	5	40	Weak	
2299-03	FAR HW	105.9	13260	8.430	7.538	180.9	AGRP	chl	10	sil	5	py	3	18	Weak	
2299-04	FAR HW	107.3	7400	4.705	4.207	101.0	AGRP	chl	10	sil	5	py	3	18	Weak	
2299-05	FAR HW	109.1	13000	8.265	7.391	177.4	AGRP	chl	10	sil	5	py	3	18	Weak	
2299-06	IMM HW	110.5	10320	6.561	5.867	140.8	HPGD	clay	35	hem	30	sil	12	94	Strong	10% chl/7% lim/qtz segs /mineralized zones
2299-07	IMM HW	111.1	8200	5.213	4.662	111.9	HPGD	clay	35	hem	30	sil	12	94	Strong	10% chl/7% lim/qtz segs /mineralized zones
2299-08	IMM HW	111.7	2940	1.869	1.671	40.1	HPGD	clay	35	hem	30	sil	12	94	Strong	10% chl/7% lim/qtz segs /mineralized zones
2299-09	IMM HW	112.3	8580	5.455	4.878	117.1	HPGD	clay	35	hem	30	sil	12	94	Strong	10% chl/7% lim/qtz segs /mineralized zones
2299-10	IMM HW	112.8	9160	5.824	5.208	125.0	HPGD	clay	35	hem	30	sil	12	94	Strong	10% chl/7% lim/qtz segs /mineralized zones
2299-11	ORE1	113	1580	1.005	0.898	21.6	HPGD	clay	35	hem	30	sil	12	94	Strong	10% chl/7% lim/qtz segs /mineralized zones
2299-12	ORE1	113.2	80	0.051	0.045	1.1	HPGD	clay	35	hem	30	sil	12	94	Strong	10% chl/7% lim/qtz segs /mineralized zones
2299-13	ORE1	113.4	720	0.458	0.409	9.8	HPGD	clay	35	hem	30	sil	12	94	Strong	10% chl/7% lim/qtz segs /mineralized zones
2299-14	HW/FW	113.9	940	0.598	0.534	12.8	HPGD	clay	35	hem	30	sil	12	94	Strong	10% chl/7% lim/qtz segs /mineralized zones
2299-15	HW/FW	114.4	7300	4.641	4.150	99.6	HPGD	clay	35	hem	30	sil	12	94	Strong	10% chl/7% lim/qtz segs /mineralized zones
2299-16	HW/FW	114.7	1960	1.246	1.114	26.7	HPGD	clay	35	hem	30	sil	12	94	Strong	10% chl/7% lim/qtz segs /mineralized zones
2299-17	ORE2	115.2	2000	1.272	1.137	27.3	HPGD	clay	35	hem	30	sil	12	94	Strong	10% chl/7% lim/qtz segs /mineralized zones
2299-18	ORE2	115.7	2960	1.882	1.683	40.4	HPGD	clay	35	hem	30	sil	12	94	Strong	10% chl/7% lim/qtz segs /mineralized zones
2299-19	ORE2	116.1	3920	2.492	2.229	53.5	HPGD	clay	35	hem	30	sil	12	94	Strong	10% chl/7% lim/qtz segs /mineralized zones
2299-20	ORE2	116.5	0	0.000	0.000	0.0	HPGD	clay	35	hem	30	sil	12	94	Strong	10% chl/7% lim/qtz segs /mineralized zones
2299-21	ORE2	116.8	1540	0.979	0.876	21.0	AGRP	chl	10	sil	7	py	5	22	Weak	
2299-22	FW	117.2	1860	1.183	1.057	25.4	AGRP	chl	10	sil	7	py	5	22	Weak	
2299-23	FW	117.8	2720	1.729	1.546	37.1	AGRP	chl	10	sil	7	py	5	22	Weak	
2299-24	FW	118.5	4860	3.090	2.763	66.3	AGRP	chl	10	sil	7	py	5	22	Weak	
2299-25	FW	119	2580	1.640	1.467	35.2	AGRP	chl	10	sil	7	py	5	22	Weak	
2602 tested June 13, 2005																
2602-01	FAR HW	93.2	12220	7.769	6.947	166.7	AGRP	chl	12	sil	5	clay	5	22	Weak	
2602-02	FAR HW	95.6	6000	3.815	3.411	81.9	AGRP	chl	12	sil	5	clay	5	22	Weak	
2602-03	FAR HW	97	4860	3.090	2.763	66.3	AGRP	chl	12	sil	5	clay	5	22	Weak	
2602-04	FAR HW	98.4	12480	7.934	7.095	170.3	AGRP	chl	12	sil	5	clay	5	22	Weak	
2602-05	FAR HW	99.8	11740	7.464	6.674	160.2	AGRP	chl	12	sil	5	clay	5	22	Weak	
2602-06	IMM HW	100.8	3340	2.123	1.899	45.6	AGRP	chl	12	sil	5	clay	5	22	Weak	
2602-07	IMM HW	101.6	9460	6.014	5.378	129.1	AGRP	chl	12	sil	5	clay	5	22	Weak	
2602-08	IMM HW	102.3	1200	0.763	0.682	16.4	AGRP	chl	12	sil	5	clay	5	22	Weak	
2602-09	IMM HW	103.2	4680	2.975	2.661	63.9	AGRP	clay	25	chl	17	sil	7	49	Weak	
2602-10	IMM HW	103.8	600	0.381	0.341	8.2	AGRP	clay	25	chl	17	sil	7	49	Weak	
2602-11	ORE	104	3080	1.958	1.751	42.0	AGRP	clay	25	chl	17	sil	7	49	Weak	
2602-12	ORE	104.3	1760	1.119	1.001	24.0	AGRP	clay	25	chl	17	sil	7	49	Weak	
2602-13	ORE	104.5	4700	2.988	2.672	64.1	AGRP	clay	25	chl	17	sil	7	49	Weak	
2602-14	FW	105.1	1520	0.966	0.864	20.7	AGRP	clay	25	chl	17	sil	7	49	Weak	
2602-15	FW	105.6	4380	2.785	2.490	59.8	AGRP	clay	25	chl	17	sil	7	49	Weak	
2602-16	FW	106.1	4800	3.052	2.729	65.5	AGRP	clay	25	chl	17	sil	7	49	Weak	
2602-17	FW	106.9	4640	2.950	2.638	63.3	AGRP	clay	25	chl	17	sil	7	49	Weak	
2602-18	FW	107.8	800	0.509	0.455	10.9	AGRP	clay	25	chl	17	sil	7	49	Weak	

Appendix B

“Rabbit Lake Mine
Converting Geology Classification to Rock Classification”
By Sutton and Milne (1998)

1998.

Rabbit Lake Mine
Converting Geology Classification to Rock Classification

There is a large amount of geological data collected concerning the alteration and strength of the rock. If this data can be converted to Q' classification values used in stope design, it will make it much easier to predict stope hanging wall and back stability.

The geology department assesses the rock mass in terms of the degree of alteration and the rock strength and uses the following categories:

Alteration Code	Description	% of rock mass altered
A1	Fresh	0%
A3	Weakly Altered	0-5%
A5	Moderately Altered	5-25%
A7	Strongly Altered	25-100%

Rock Strength Code	Description
R1	Very weak rock – Indents with thumbnail Can be dug out with rock hammer
R2	Weak rock – can be peeled with a knife Can be dented with a rock hammer
R3	Medium strong rock – requires several blows with a rock hammer to break

Based on comments from geology staff, it appears that a weak rock, R1, may have significant alteration whereas a highly altered rock may still have significant strength, R2 or R3.

The following parameters need to be assessed to obtain a Q' classification value:

- RQD - Based on degree of fracturing and shearing
- Jn - Based on number of joint sets presents
- Jr - Roughness of the joint surfaces
- Ja - Alteration of joint surfaces

Based on mapping done by D. Sutton and D. Milne, it appears that the rock strength code, R1 to R3, may have the most significant influence on the Q' classification value. The A7 and A3 alteration codes primarily act to influence the joint classification rating in the stronger R2 and R3 categories. The R1 and R3 categories should have the following general classification parameters:

R1 – Very Weak Rock

- RQD, (10% to 50%)

Significant zones of lost, crushed or dented core would be expected in an R1 rock, reducing the RQD. Intact rock which could easily be crumbled by hand should be given a low (10%) RQD.

- Jn, (4)

R1 zones occur in conjunction with shearing and only the foliation joint set is easily observed. Two joint sets may be apparent in R1 zones.

- Jr (.75)

Joint surfaces are often wavy, but smooth/polished or slickensided. A Jr value of 1.5 to 2.0 can be assigned however, this value should be divided by 2 if the surfaces are polished or slickensided.

- Ja (8 to 12)

This value is difficult to quantify and can range from 8 to 12 and assumes the rock on either side of a shear will not come in contact when shearing occurs.

The following classification values are approximated for R1 categories:

R1 /A3 Does not occur on site

R1 / A5 RQD - 50% (10% to 50%)
Jn - 4 (2 sets)
Jr - 0.75 (Planar to wavy & slickensided)
Ja - 8 (8 to 12)
Q' = 1.2

R1 / A7 RQD - 20% (10% to 50%)
Jn - 4 (2 joint sets)
Jr - 0.75 (Planar to wavy & slickensided)
Ja - 10 (8 to 12)
Q' = 0.4

R2 – Weak Rock – Breaks with one blow

- RQD, (60% to 90%)

A significant amount of the rock mass consists of rock that can be dented with a pick. Core from these zone could be easily broken by had giving a low (10%) RQD.

- Jn, (6 to 9)

Two plus random to three joint sets are commonly seen in this rock mass category.

- Jr, (1.5)

Joints are commonly planar to wavy. A higher Jr value could be encountered in the pegmatite.

- Ja, (4.0 to 8.0)
- Significant alteration is usually present on the joint surfaces, minor shearing may be present.

The following classification values are <u>approximated</u> for R2 categories:

R2 /A3 Not too common.
RQD - 90%
Jn - 9
Jr - 1.5
Ja - 4.0
Q' - 3.8

R2 / A5 RQD - 75%
Jn - 6 to 9
Jr - 1.5
Ja - 6.0
Q' - 2.5

R2 / A7 RQD - 60%
Jn - 6
Jr - 1.5
Ja - 6
Q' - 2.5

<u>R3 – Medium Strong Rock – Many blows to break</u>

- RQD, (90% to 100%) Competent rock.
- Jn, (9) Three joint sets are generally apparent.
- Jr, (1.5 to 2.3) Joints are generally slightly rough in the foliated rocks, however, if the pegmatite the joints are more rough and between planar & wavy.
- Ja (1.0 to 1.5) There is biotite on the foliated rock and the no alteration on the pegmatite joints, unless the rock mass is altered.

The following values are <u>approximated</u> for the foliated rocks:

R3 /A3 Not too common.
RQD - 100%, 100
Jn - 9, 6 (3 joint sets)
Jr - 1.5, 3.0 (slightly rough)
Ja - 1.5, 1.5 (trace mica on surfaces)
Q' - 11

R3 / A5 RQD – 100%
Jn - 9
Jr - 1.5
Ja - 2.0 (Increased alteration on jnt surfaces)
Q' - 8.3

R3 / A7 RQD - 90%
Jn - 9
Jr - 1.5
Ja - 4.0
Q' - 3.8

The following values are <u>approximated</u> for the pegmatoid rocks:

R3 /A3 RQD - 100%
Jn - 9 (3 joint sets)
Jr - 2.3, (slightly rough)
Ja - 1.0 (trace mica on surfaces)
Q' - 25

R3 / A5 RQD – 100%
Jn - 9
Jr - 2.3
Ja - 2.0 (Increased alteration on jnt surfaces)
Q' - 13

R3 / A7 Not too common
RQD – 90%
Jn - 9
Jr - 12.3
Ja - 4.0
Q' - 5.8

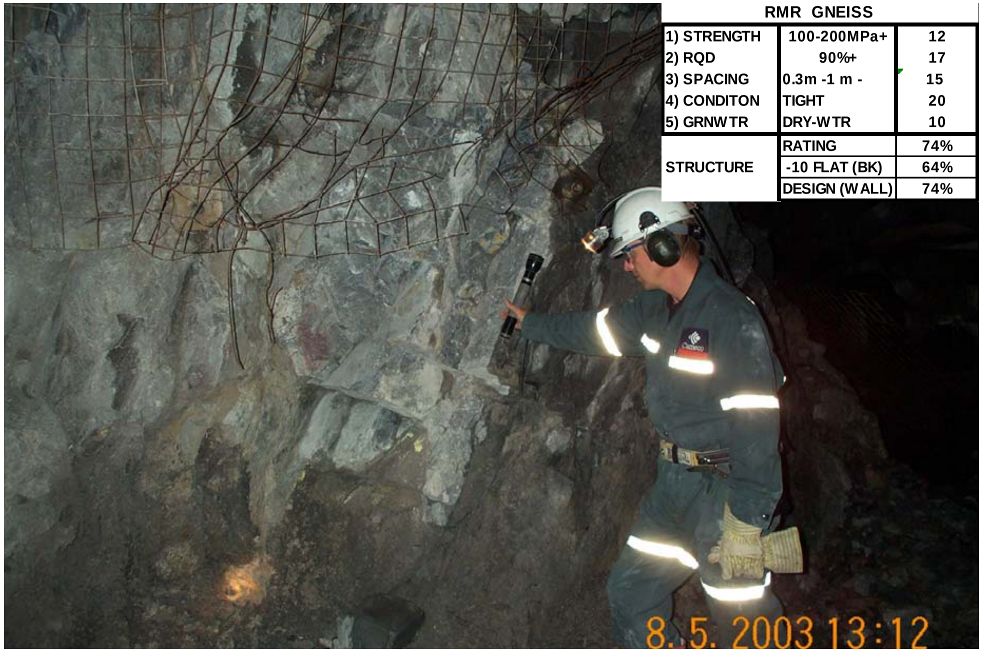
It should be noted that these Q' classification values are only preliminary and will change. Some of the alteration/strength categories have not yet been mapped for Q' values and the parameters have been inferred. Also, estimates for the Ja, joint alteration parameters, tends to be subjective for Ja values greater than 4.0. Work will continue for the estimation of the Q' classification values from the alteration/hardness mapping through continued underground mapping using both systems. Stope assessments of dilution will also be used to back analyse Q' values, within a reasonable range.

The goal of this work is to document stope hanging wall dilution using the Potvin Stability Graph and the Dilution graph to determine if past hanging wall dilution values can be predicted based on stope geometry and rock mass conditions. If this work is successful, it should be possible to design stope geometries to which do not result in hanging wall dilution exceeding a predetermined maximum.

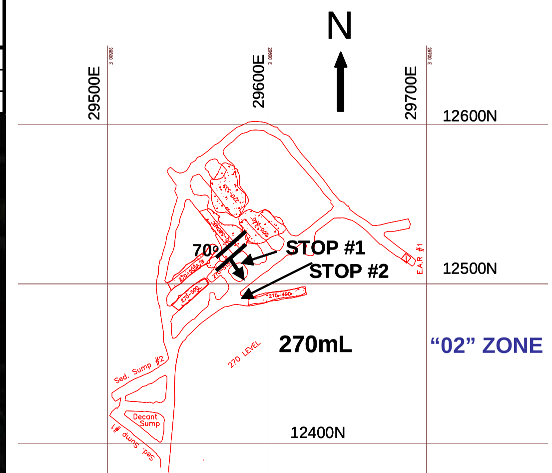
Dan Sutton Doug Milne

Appendix C

Rock Mass Classification Observations from
External Rock Mechanics Audit Reports



RMR GNEISS		
1) STRENGTH	100-200MPa+	12
2) RQD	90%+	17
3) SPACING	0.3m -1 m -	15
4) CONDITON	TIGHT	20
5) GRNWTR	DRY-WTR	10
STRUCTURE	RATING	74%
	-10 FLAT (BK)	64%
	DESIGN (WALL)	74%



PLAN OF 270mL

STOP #1: 270L, 290-505 O/C

TYPICAL HW IN "02" ZONE. RMR OF WALL = 74%, BACK DUE TO FLAT STRUCTURES 64%. SPAN = 5m. IMMEDIATE HW IS FRESH GNEISS. DOMINANT JS/FOLIATION DDR=140º DIP=70º.

STRIKE LENGTH = 20m , DIP LENGTH = 23m RESULTS IN HR=5.3m

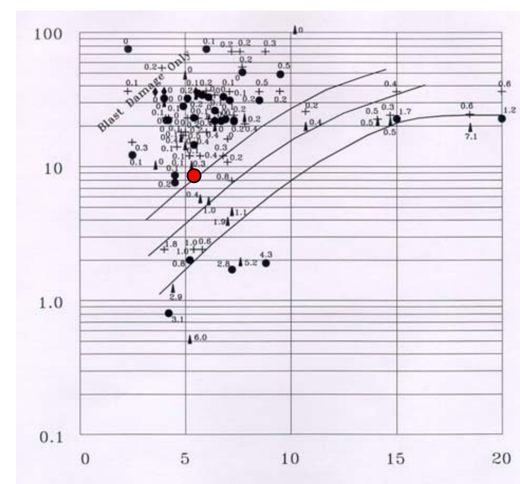
Q = (90/9) * (1.5/4) = 3.75 Ja=4 DUE TO PRESENCE OF FL WITH POTENTIALLY GRAPHITE/CLAY (J.R.)

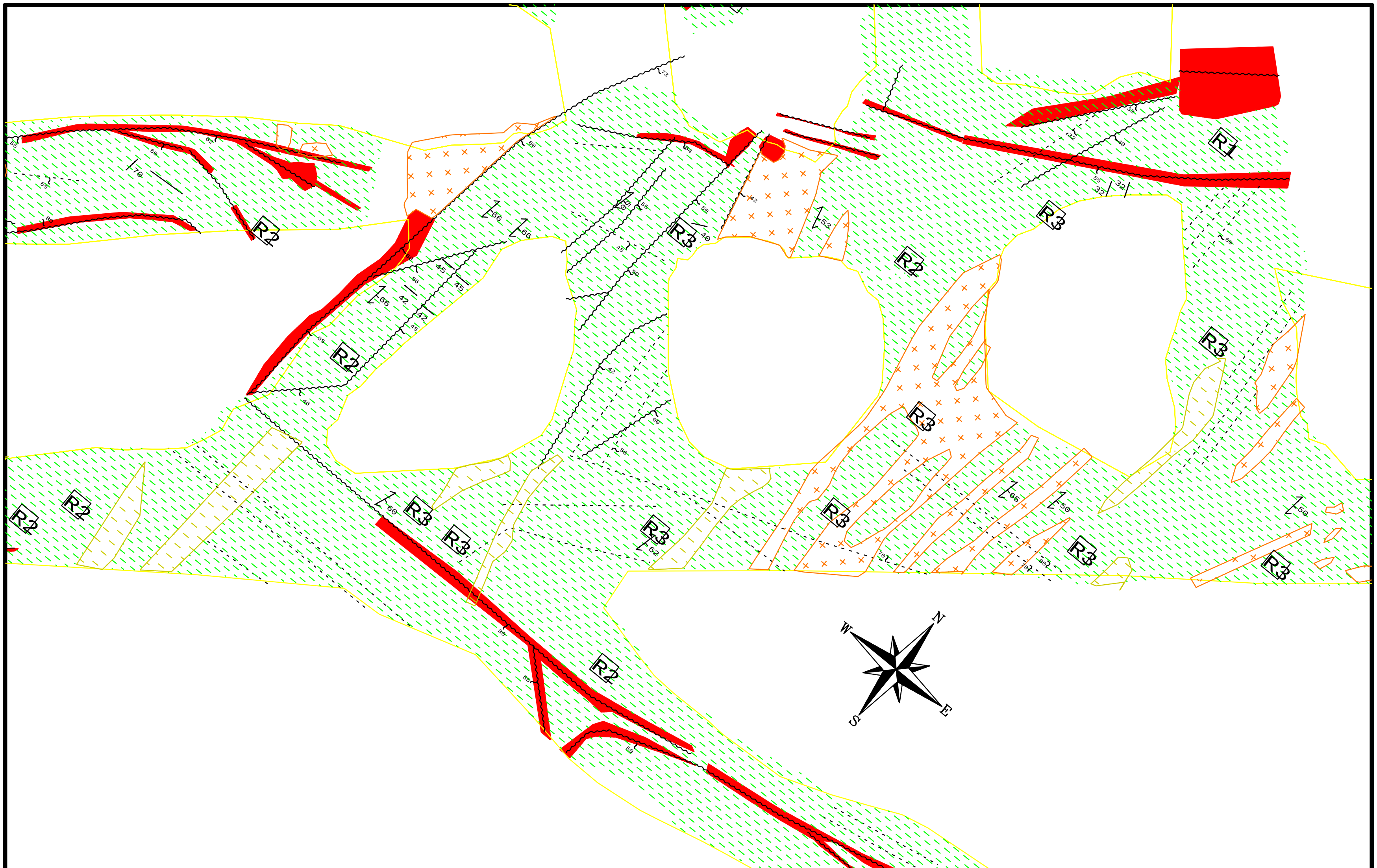
N' = 3.75 * 1 * 0.4 * 5.2 = 7.8

B = 0.4 SINCE 30º to 40º DIFFERENCE IN STRIKE AND 20º IN DIP

C = 5.2 DUE TO 62º AVG DIP HW

STOPE YET TO BE MINED IS BEING DEVELOPED





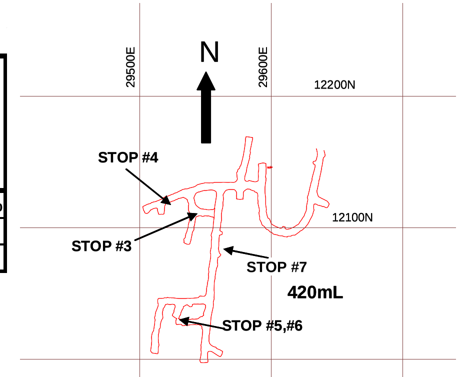
RABBIT LAKE OPERATION
EAGLE POINT MINE

TITLE:

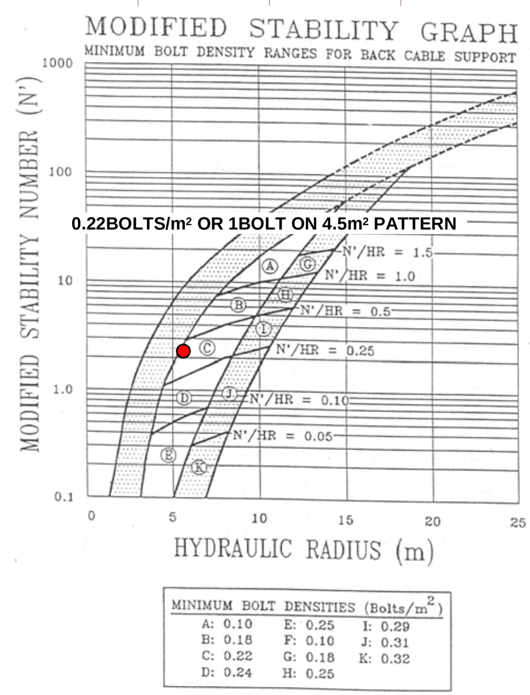
EAGLE POINT MINE
270L GEOLOGICAL MAPPING

DATE:	23-Nov-2010	DWG #:
SCALE:	1:250	REVISION:
BY:	KF	FILE:
REVISED BY:		

RMR HW		
1) STRENGTH	10-25MPa	2
2) RQD	50%+	8
3) SPACING	50-300mm	10-15
4) CONDITON	GRPH/CLAY	6
5) GRNWTR	DRY-WTR	10
STRUCTURE	RATING	36% - 41%
	DESIGN (WALL)	40%



PLAN OF 420mL



STOP #3: 420L, 420-060 U/C

TYPICAL BAD HW HAVING AN RMR OF 40%. SUPPORT IS 1.2m X 1.2m (4X4ft), 2.4m (8ft) LONG REBAR IN WALLS + SPLITS AT BASE. WHERE NO BOLTS AT BASE OF WALL (BOTTOM 0.9m) SHOTCRETE FAILED/SLOUGHED (WALL MOVEMENT/SCOOP TRAM/NO SUPPORT). BACK SPAN = 4.5m WITH 40% RMR IS STABLE DUE TO SUPPORT IN PLACE.

STRIKE LENGTH = 30m , DIP LENGTH = 20m RESULTS IN HR=6m

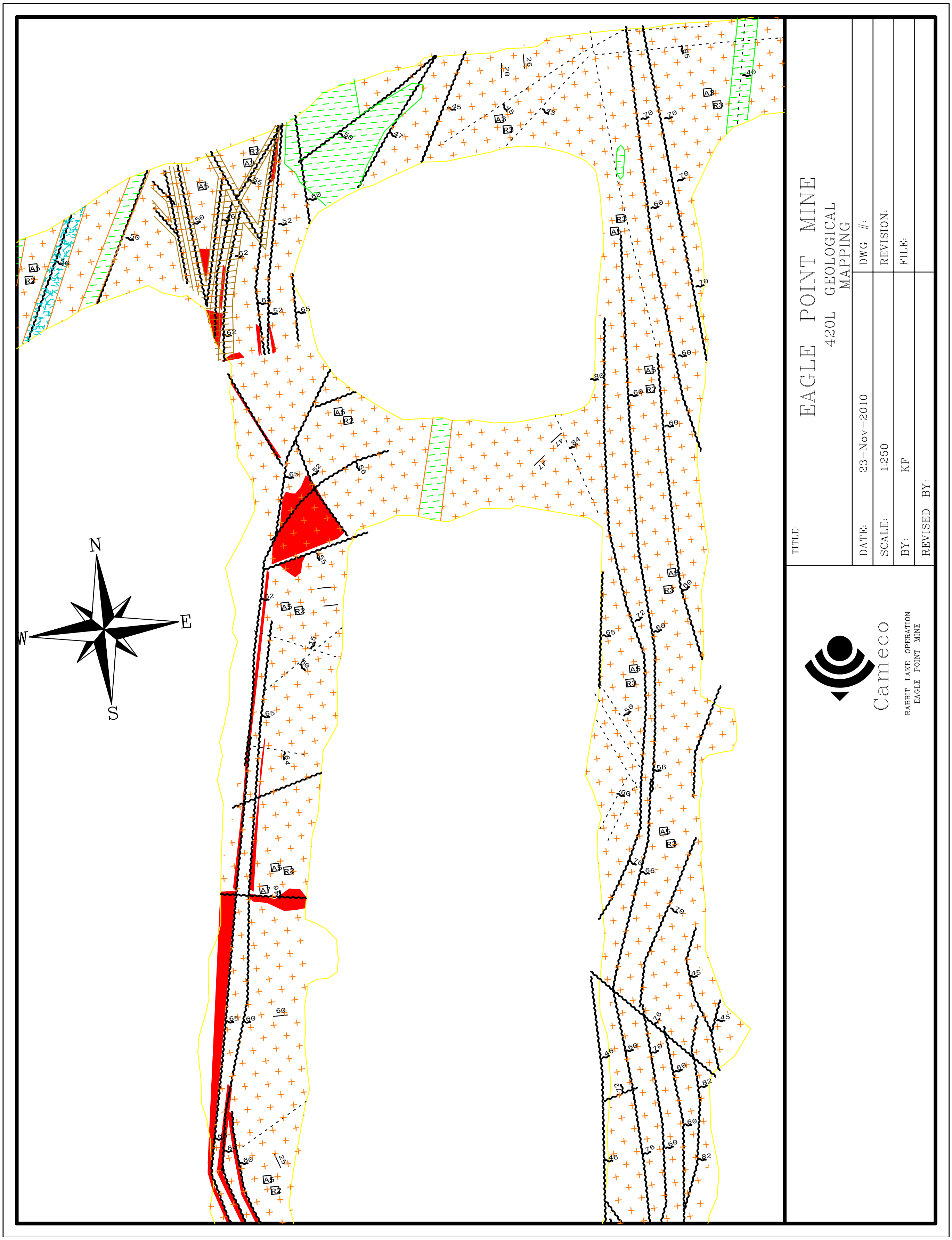
Q = (50/6) * (1.5/6) = 2.1

N' = 2.1 * 1 * 0.2 * 5 = 2.1

B = 0.2 SINCE X-CUTTING AT 20º DIP

C = 5. DUE TO 60º AVG DIP HW

STOPE YET TO BE MINED IS BEING DEVELOPED. CABLING IS 2m X 2.5m PATTERN FROM HW DRIFT OR 1 BOLT / $5m^2$.


EAGLE POINT MINE
420L GEOLOGICAL MAPPING
TITLE:
DWG #:
REVISION:
FILE:
DATE: 23-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
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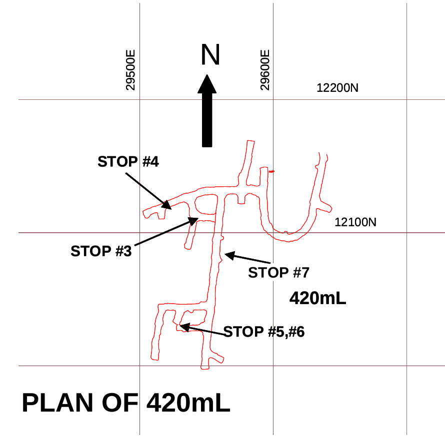
RMR HW		
1) STRENGTH	50-100MPa	7+
2) RQD	50%-75%	13
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT	20-12
5) GRNWTR	DRY-WTR	10
	RATING	65% - 57%
STRUCTURE		
	DESIGN (WALL)	60%



PLAN OF 420mL

STOP #4: 420L, #1 X-CUT BETWEEN 060 ACCESS/420-075U/C.

LONGITUDINAL BETWEEN HW AND FW LENSES. TYPICAL MODERATE HW HAVING AN RMR OF 60% DESIGN FOR HW. PEGMATITE HW.

STRIKE LENGTH = 30m , DIP LENGTH = 20m RESULTS IN HR=6m

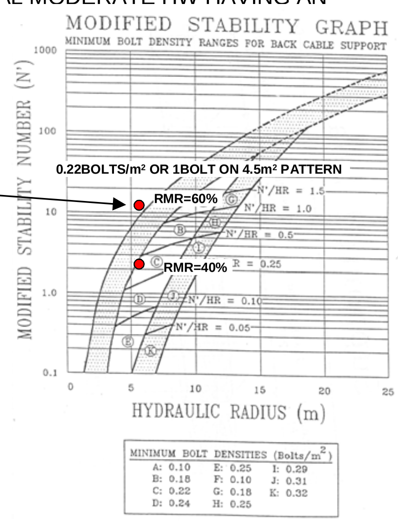
Q = (75/6) * (1.5/1.5) = 12.5 (~67% RMR equivalent.)

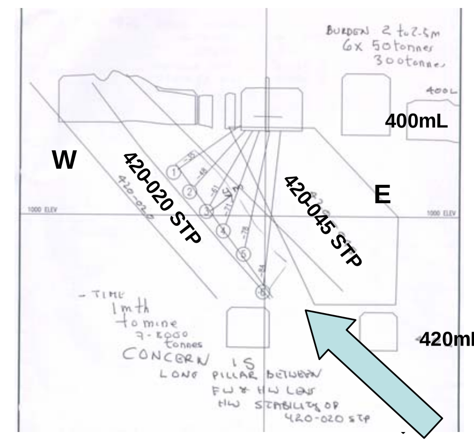
N' = 12.5 * 1 * 0.2 * 5 = 12.5

B = 0.2 SINCE X-CUTTING AT 20º DIP

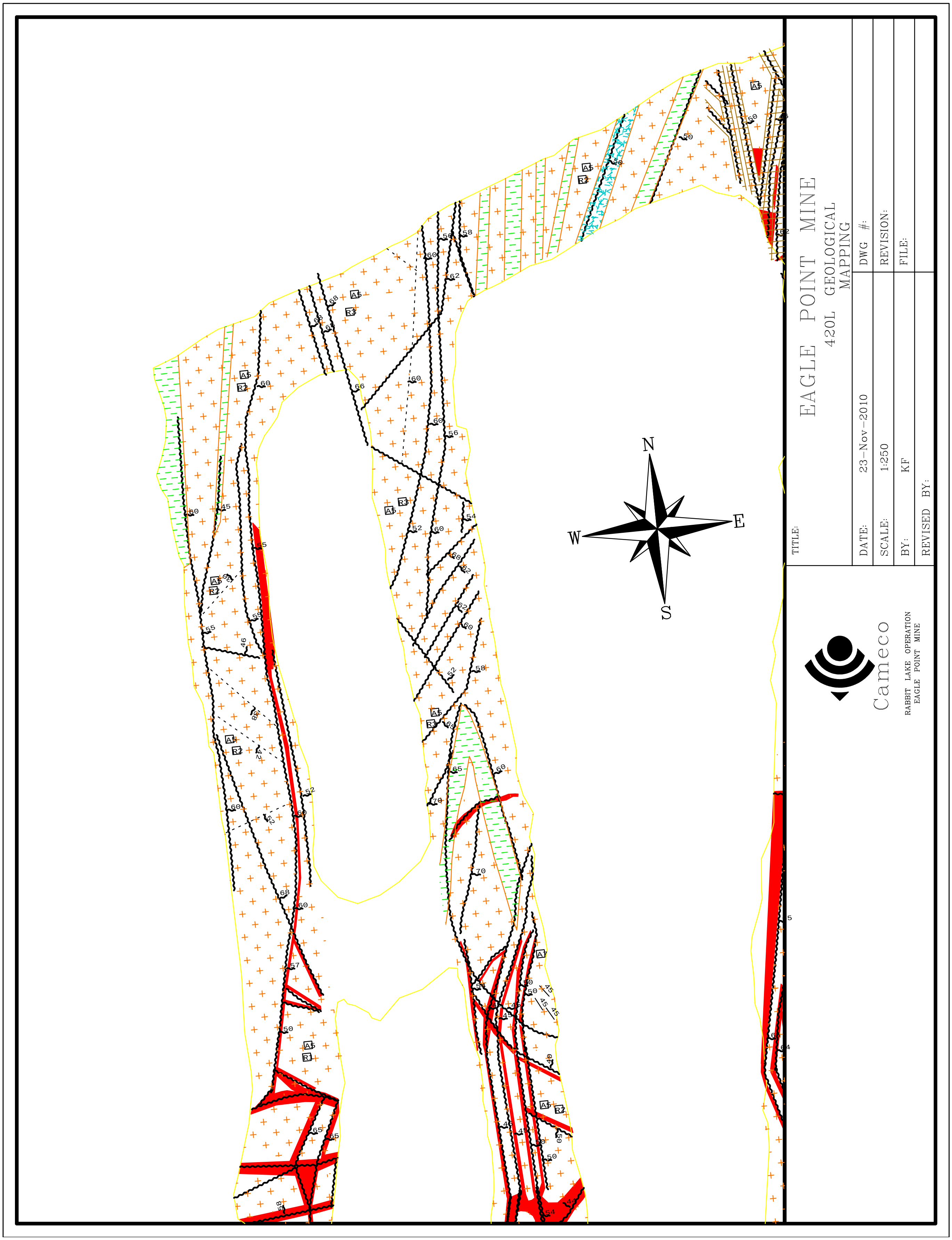
C = 5. DUE TO 60º AVG DIP HW

STOPE YET TO BE MINED IS BEING DEVELOPED. CABLING IS 2m X 2.5m PATTERN FROM HW DRIFT OR 1 BOLT / 5m².





VERTICAL SECTION LOOKING NTH THRU 020FW STOPE AND 045 STOPE.
SIMILAR TO 420-060 STP AND 075STP.


EAGLE POINT MINE
420L GEOLOGICAL MAPPING
TITLE:
DATE: 23-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
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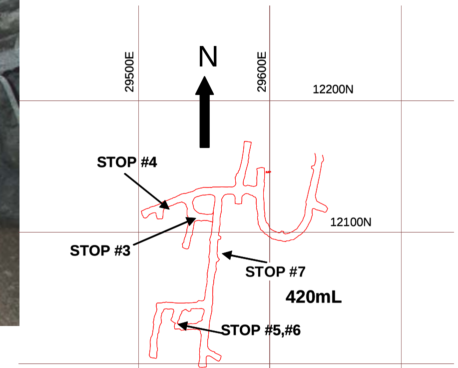
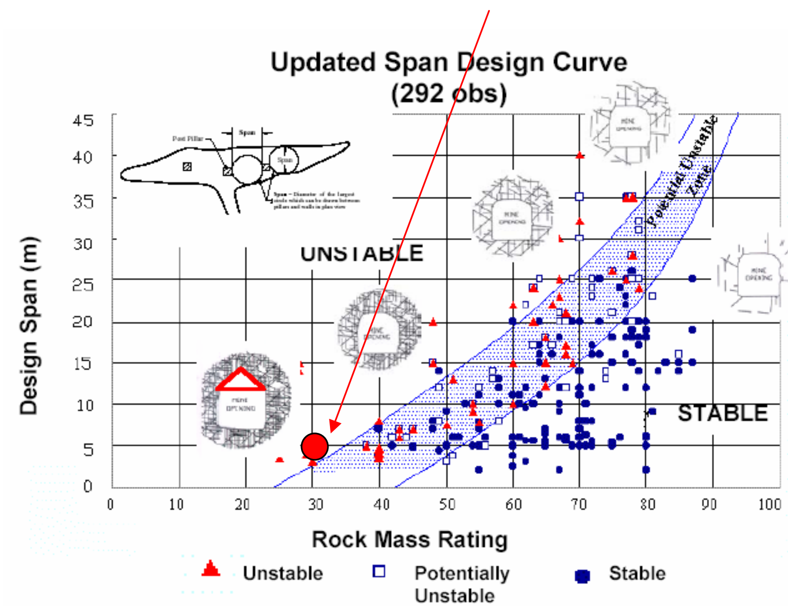


PLAN OF 420mL

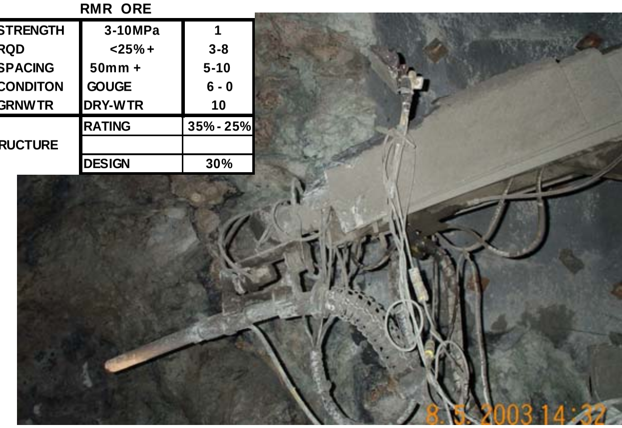
STOP #5: 420L, 420-045 U/C

LOWEST RMR FOR ORE – "WORST CASE". PITCHBLEND ORE – HI GRADE. RMR OF ORE IS 30%.THE AREA IS MOIST YIELDING A WEAKER ROCK MASS. FLOOR OF DRIVE IS CONCRETE DUE TO GAMMA CONTROL. FACE READY FOR SHOTCRETE. BACK SPAN IS 5m. MINIMUM OF 76mm (3") SHOTCRETE PLACED DUE TO GAMMA CONTROL.. SCHMIDT HAMMER USED TO TEST SHOTCRETE YIELDING INDEX TESTS IN EXCESS OF 30MPa UCS.



RMR ORE

1) STRENGTH	3-10MPa	1
2) RQD	<25% +	3-8
3) SPACING	50mm +	5-10
4) CONDITON	GOUGE	6 - 0
5) GRNWTR	DRY-WTR	10
STRUCTURE	RATING	35% - 25%
	DESIGN	30%



REMOTE SHOTCRETE MACHINE.

EAGLE POINT MINE
420L GEOLOGICAL MAPPING
TITLE:
DWG #:
REVISION:
FILE:
DATE: 23-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
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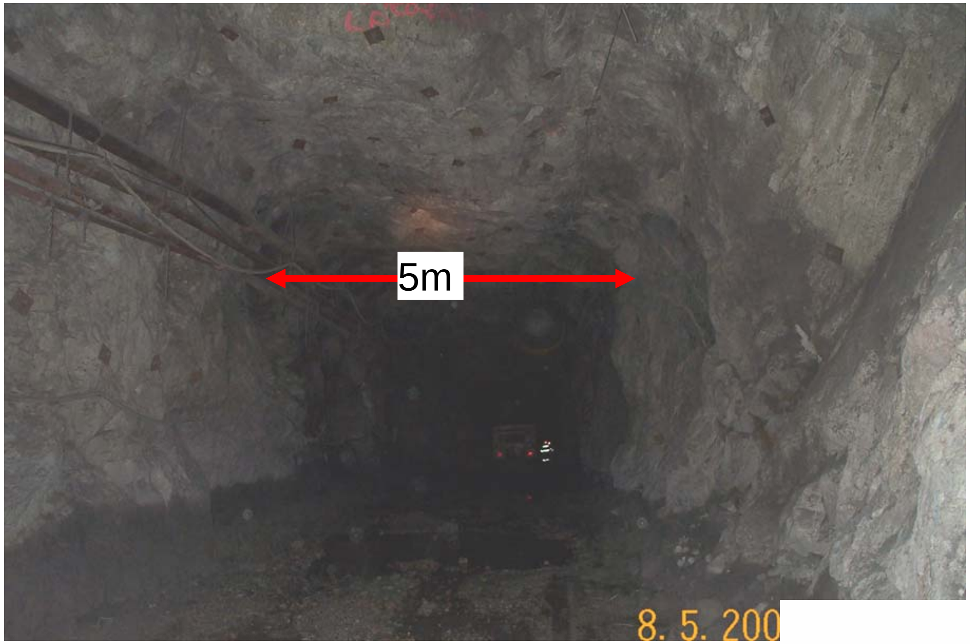
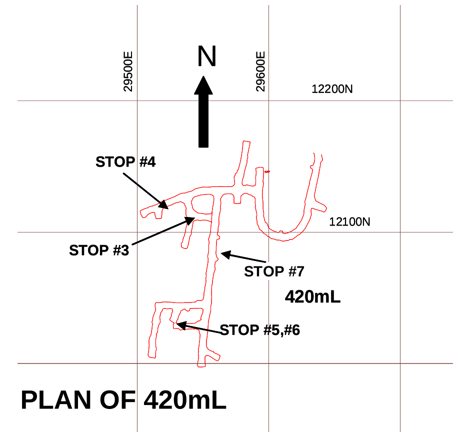
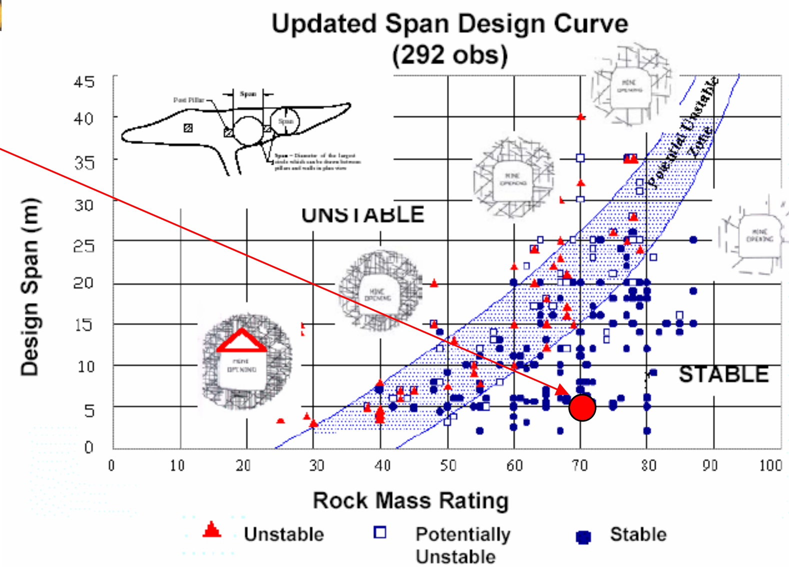


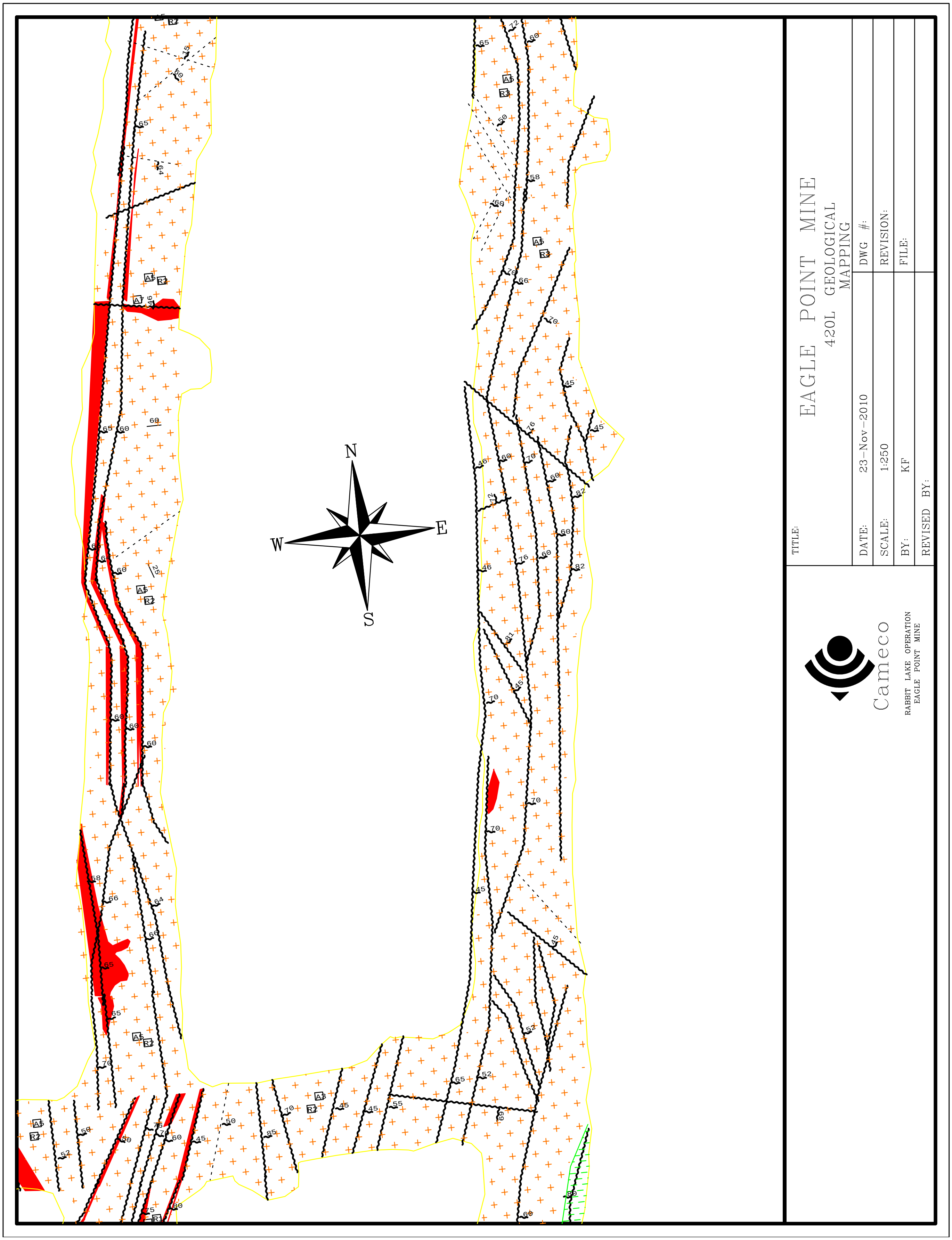


STOP #7: 420L MAIN HAULAGE. SPAN 5m(16ft) X 4.5m (15ft) IN HEIGHT. RMR OF BACK (FLAT JNTS) 70%. SUPPORT IS 2.4m (8ft) MECHANICAL BOLTS ON A 1.2m X 1.2m (4ft X 4ft) PATTERN IN BACK WITH 1.8m (6ft) MECHANICAL BOLTS IN WALLS. ADVERSE STRUCTURE NOT OBSERVED. STABLE.



RMR GNEISS/PEGMATITE

1) STRENGTH	100 - 200MPa	12
2) RQD	90%-100%	17
3) SPACING	0.3m - 1m+	20+
4) CONDITON	TIGHT	20
5) GRNWTR	DRY-WTR	10
	RATING	79%+
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	70%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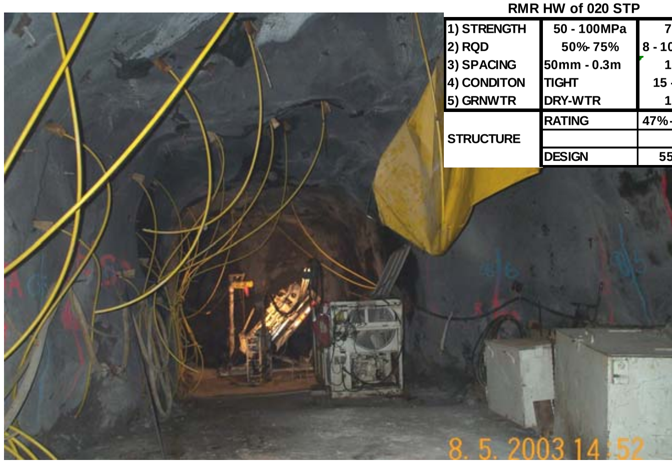
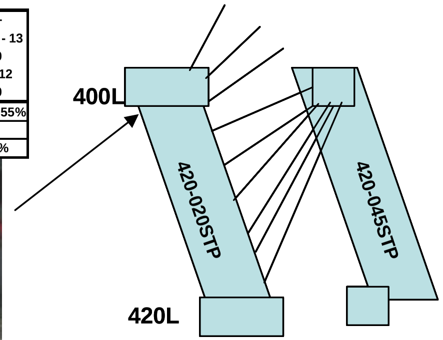
TITLE:
EAGLE POINT MINE
420L GEOLOGICAL
MAPPING
DATE: 23-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE

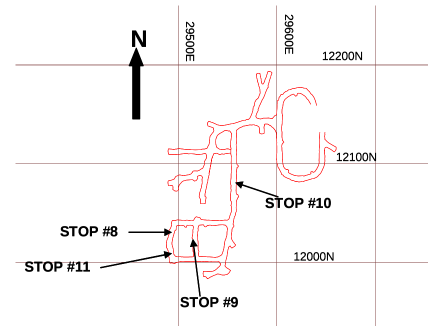


RMR HW of 020 STP

1) STRENGTH	50 - 100MPa	7+
2) RQD	50%-75%	8 - 10 - 13
3) SPACING	50mm - 0.3m	10
4) CONDITON	TIGHT	15 -12
5) GRNWTR	DRY-WTR	10
STRUCTURE	RATING	47%- 55%
	DESIGN	55%



SCHEMATIC VERTICAL
SECTION LOOKING NTH



PLAN OF 400mL

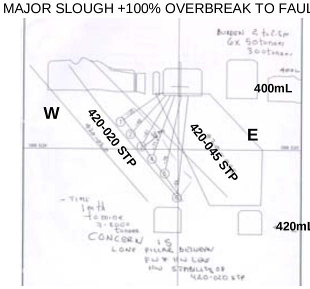
STRIKE LENGTH = 35m , DIP LENGTH = 28m RESULTS IN HR=7.8m

N' = 2.3 * 1 * 0.2 * 5 = 2.3 N' FOR 30%RMR = 0.2 X 1 X 0.2 X 5 = 0.2

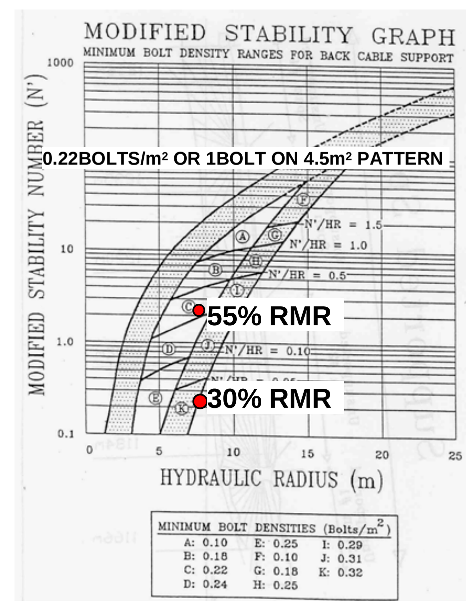
B = 0.2 SINCE 20º X CUTTING TO HW DIP

C = 4.6 DUE TO 55º AVG DIP HW

MAJOR SLOUGH +100% OVERBREAK TO FAULT +

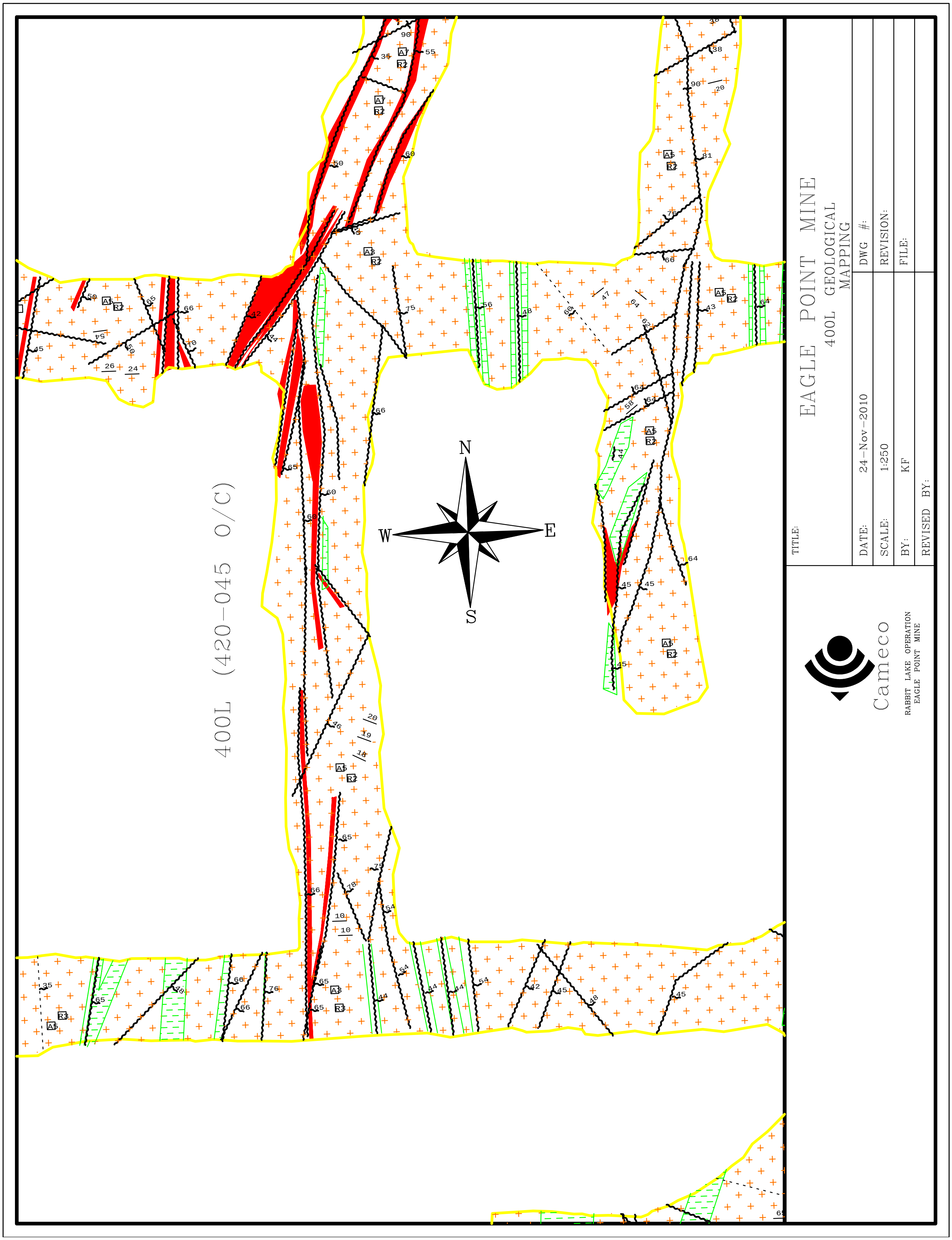


VERTICAL SECTION LOOKING NTH-
CABLE RING #12



STOP #8: 400L, 420-020 STP O/C.

CONCRETE FLOOR (GAMMA). HW OF FW LENS (420-020STP) IS CABLED TO PREVENT OVERBREAK INTO 420-020. IN ADDITION HW/BACK ON 400L U/C IS CABLED TO ARREST POTENTIAL OVERBREAK FROM MIGRATING FROM 420-400 LEVEL TO ABOVE 400mL. HW OF ORE ABOVE 400mL IS UNDERCUT ie. 380-400L. BLAST HOLES ARE 14cm (5.5") DIAM. Q'=2.3 (J.R. DESIGN). CABLING OF A 55% RMR HW WILL BE SUCCESSFUL, HOWEVER, CABLING OF A 30% RMR HW IS SUSPECT.



400L (420-045 O/C)
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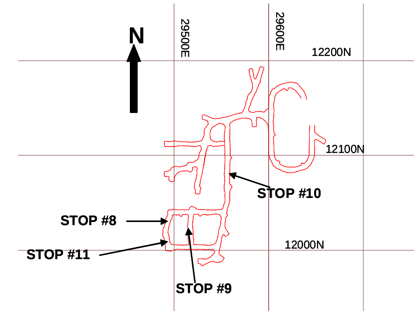
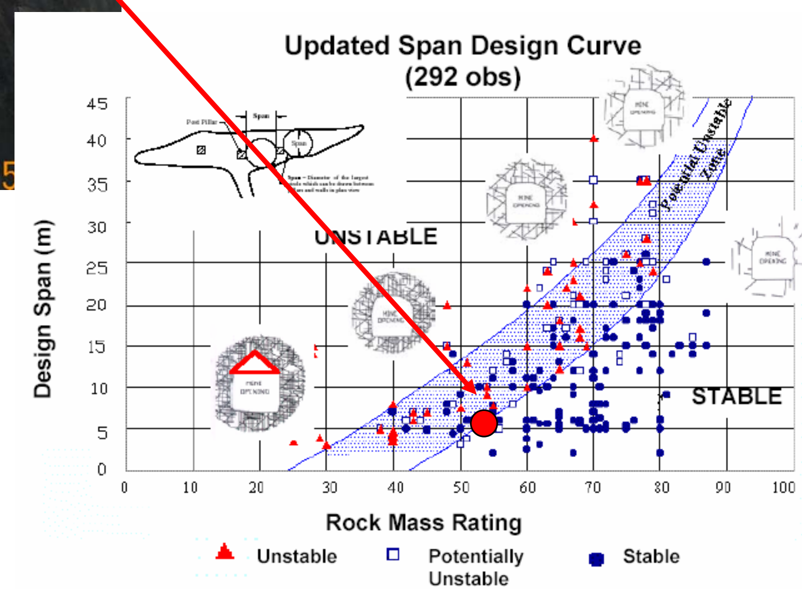
TITLE:
EAGLE POINT MINE
400L GEOLOGICAL MAPPING
DWG #:
REVISION:
FILE:
DATE: 24-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE



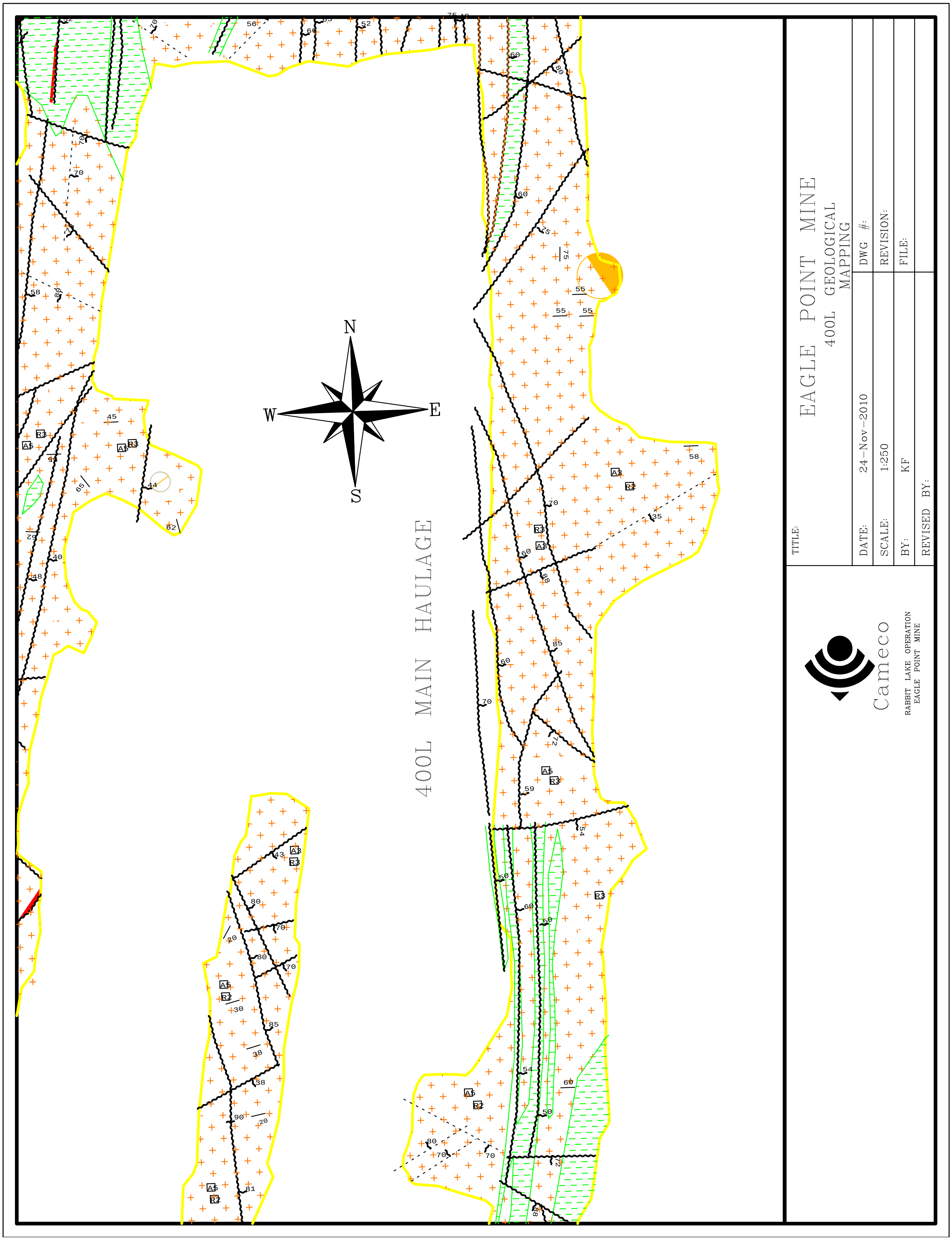
PLAN OF 400mL

STOP #10: 400L, 420-020 STP O/C HAULAGE/ACCESS

SPAN OF BACK IS 5m WITH RMR IN EXCESS OF 55%. FLAT JOINTS. MECHANICAL/REBAR IN BACK. STABLE.



	RMR BACK	
1) STRENGTH	50 - 100MPa+	7
2) RQD	50%- 75%+	15
3) SPACING	50mm - 0.3m+	15
4) CONDITON	TIGHT	15
5) GRNWTR	DRY-WTR	10
	RATING	62%
STRUCTURE	FLAT	-10%
	DESIGN	52%



EAGLE POINT MINE
400L GEOLOGICAL MAPPING
TITLE:
DWG #:
REVISION:
FILE:
DATE: 24-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
400L MAIN HAULAGE
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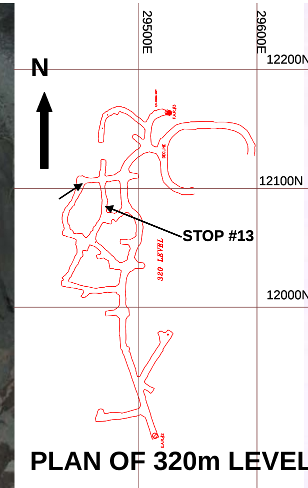
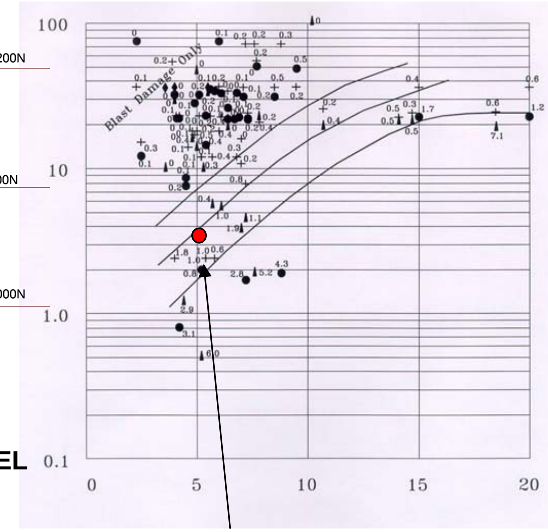
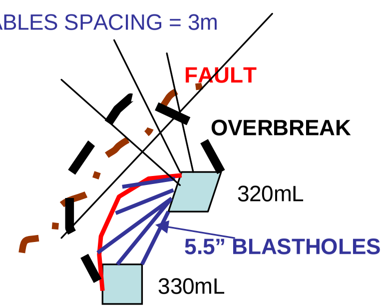




STOP #12: 320L OVERCUT, 330-090 STP O/C. LONGHOLE SLASH/SLOUGH. NOTE VERTICAL HT = 20m. 14cm (5.5”) BLASTHOLES / LONGHOLE SLASH. NO CABLES IN HW. UNDERCUT SUPPORTED. BACK SLASHED TO SPAN OF 10m+ ie. RING 6 . RMR OF BACK = 45% (FLAT JOINTS) UNSUPPORTED IN CAVE ZONE “SPAN CURVE”. NOTE CAVE ~ 0.5 * SPAN.



RMR HW/BACK

1) STRENGTH	50 - 100MPa	7
2) RQD	50%-75%	13
3) SPACING	50mm - 0.3m+	10 - 15
4) CONDITON	TIGHT/(WK)	15-12
5) GRNWTR	DRY-WTR	10
	RATING (HW)	52% - 60%
STRUCTURE	FLAT	-10%
	DESIGN(BACK)	45%

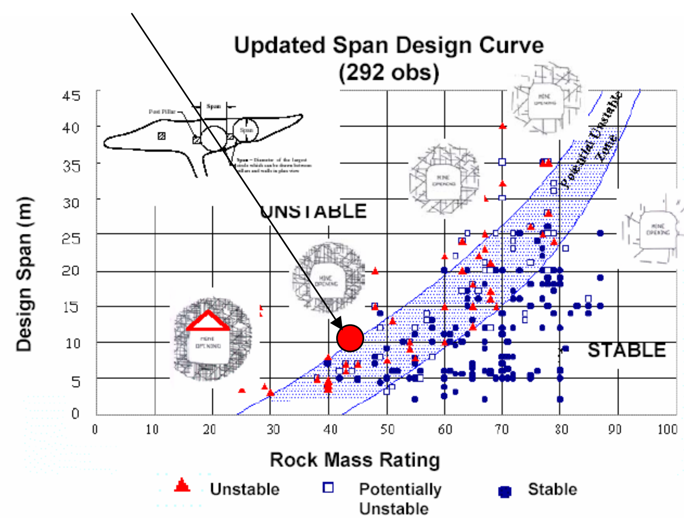
STRIKE LENGTH = 30m , DIP LENGTH = 10-15m RESULTS IN HR= 3.8 - 5m

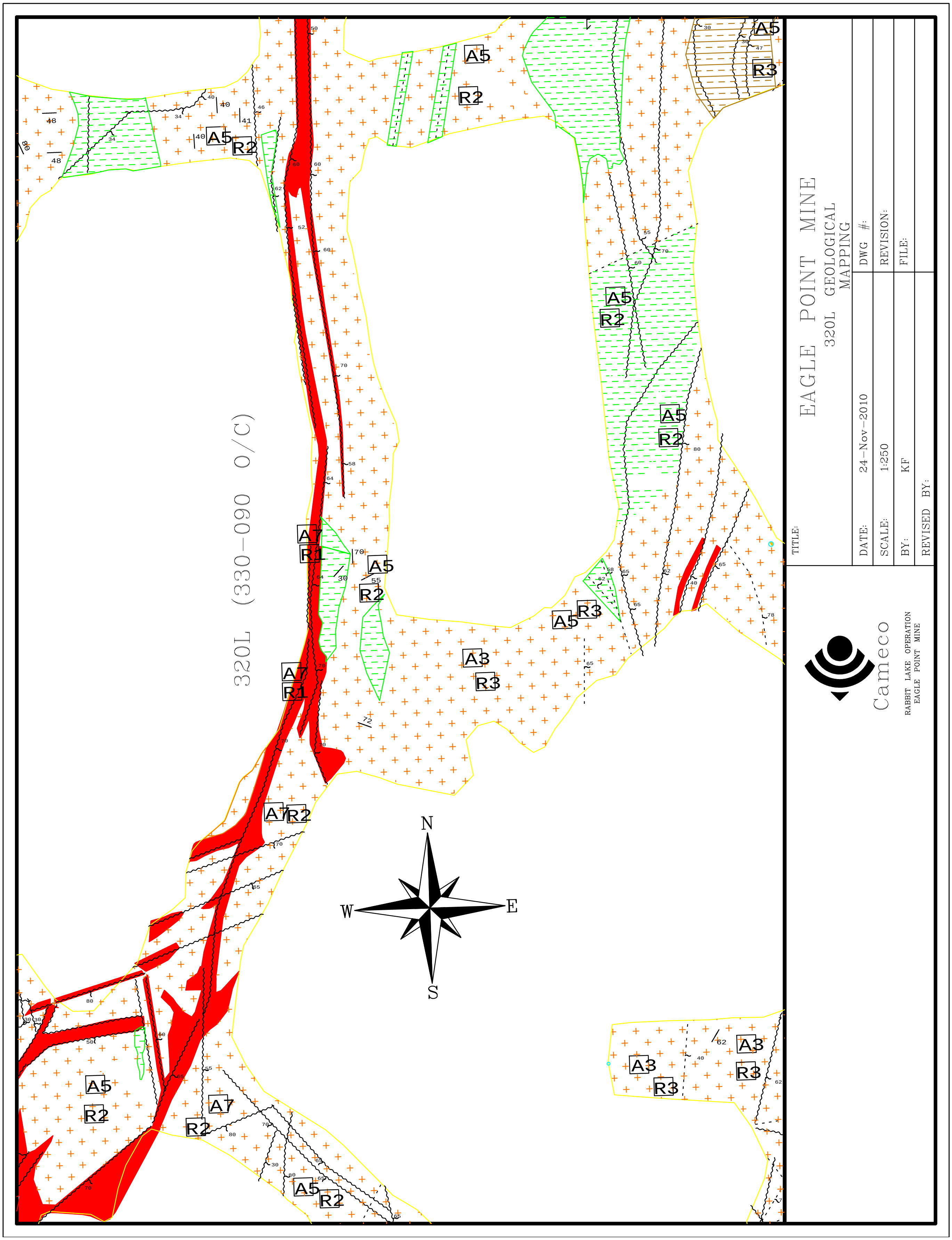
N’ = 3.4 * 1 * 0.2 * 5 = 3.4

B = 0.2 SINCE 20º X CUTTING TO HW DIP

C = 5 DUE TO 60º AVG DIP HW

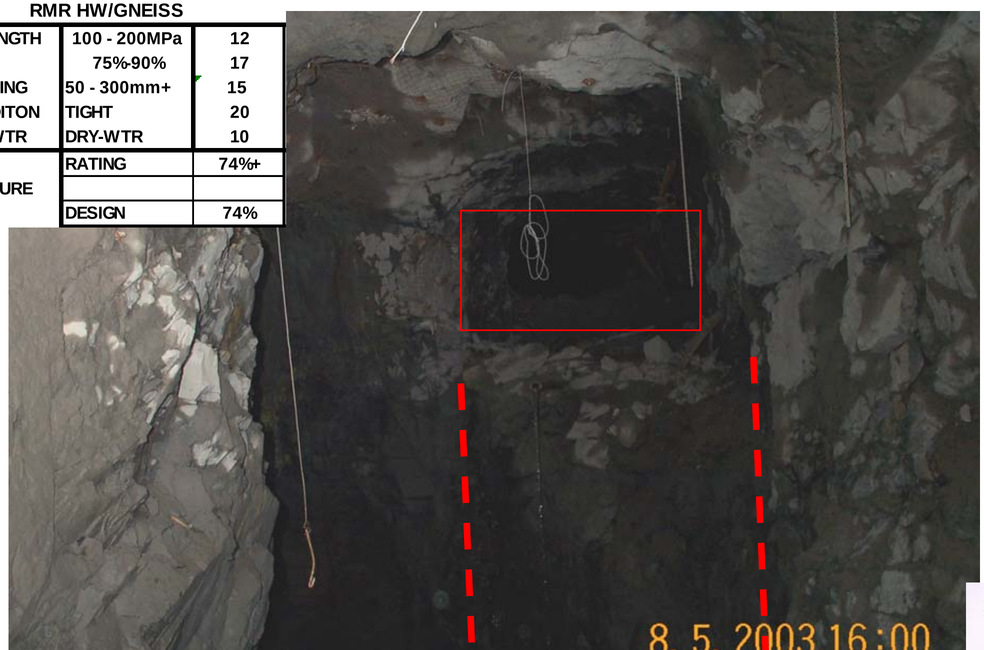
MAJOR SLOUGH +100% OVERBREAK TO FAULT +

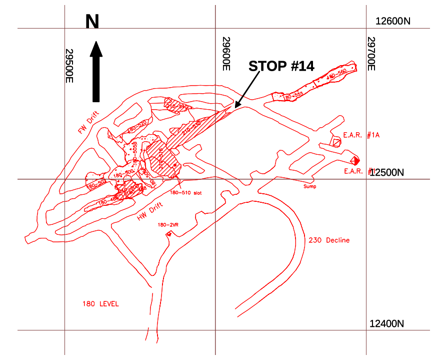



EAGLE POINT MINE
320L GEOLOGICAL MAPPING
TITLE:
DWG #:
REVISION:
FILE:
DATE: 24-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
320L (330-090 O/C)
N
E
S
W

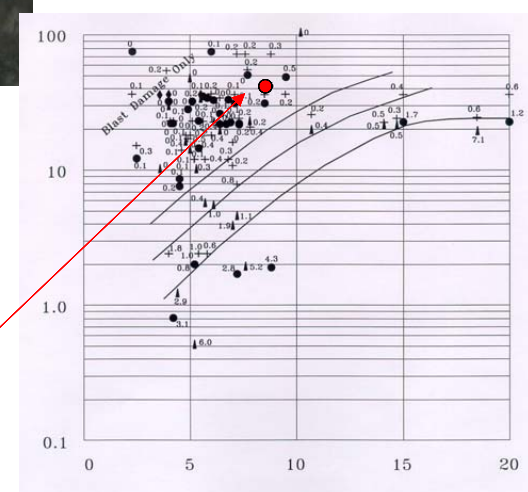
RMR HW/GNEISS

1) STRENGTH	100 - 200MPa	12
2) RQD	75%-90%	17
3) SPACING	50 - 300mm+	15
4) CONDITON	TIGHT	20
5) GRNWTR	DRY-WTR	10
	RATING	74%+
STRUCTURE		
	DESIGN	74%

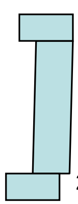




PLAN OF 180mL



STOP #14: 180 LEVEL. 210-550 STP OVERCUT. EXAMPLE OF NARROW VEIN MINING WITH EXCELLENT RMR HW. BROKE TO IMMEDIATE HW WITH MINIMAL EXTERNAL WALL SLOUGH (OVERBREAK). RMR OF HW IS 74%. MECHANICAL BOLTS ONLY USED IN BACK AND WALLS WITH SHOTCRETE. RMR OF HW AND ORE SAME – ROCK TYPE GNEISS.



STRIKE LENGTH = 35m , DIP LENGTH = 30m RESULTS IN HR=8m

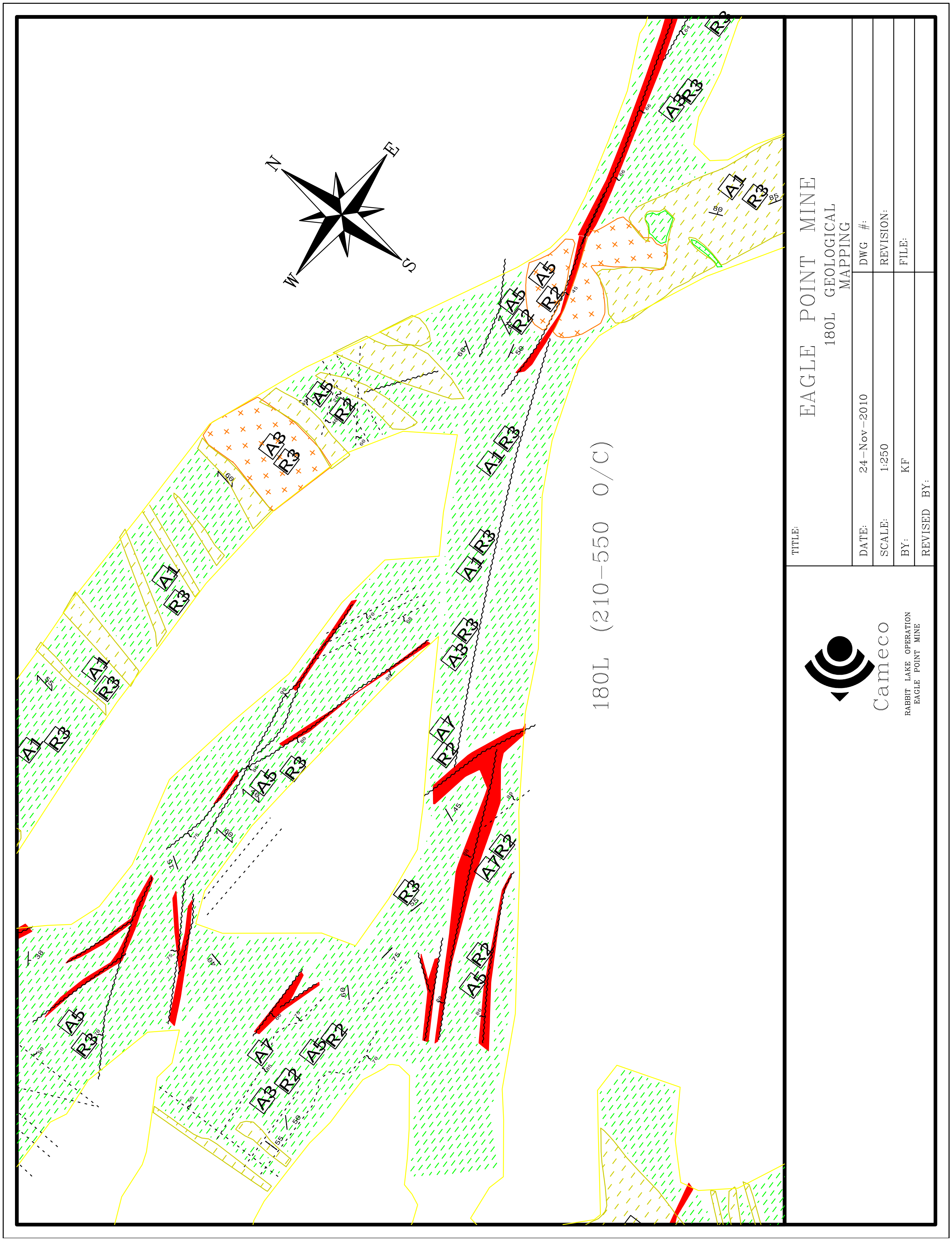
N' = 28 * 1 * 0.2 *8 = 44.8

B = 0.2 SINCE 20º X CUTTING TO HW DIP

C = 8 DUE TO 90º AVG DIP HW

STOPE MINED BLAST DAMAGE ONLY <0.5m

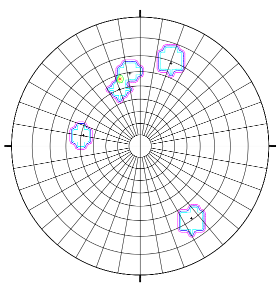
VERTICAL SCHEMATIC SHOWING UC

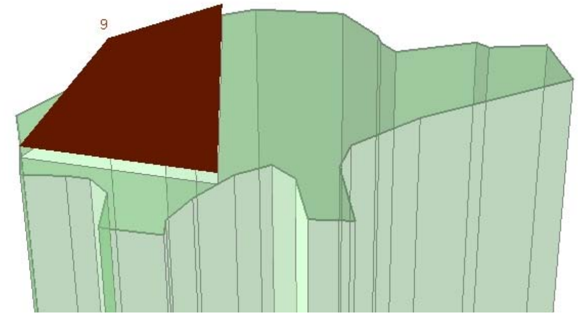


EAGLE POINT MINE
180L GEOLOGICAL
MAPPING
TITLE:
DATE: 24-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
180L (210-550 O/C)
N
E
S
W

FW OF 120-350B FW – PORPHYR FELDSPAR GNEISS ~RMR OF 54% - 45%.

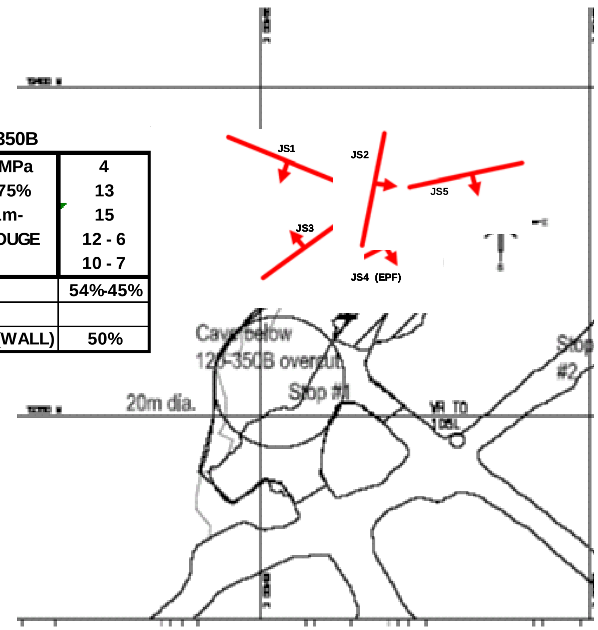
	DIP	DDR
JS1	70°	200°
JS2	50°	100°
JS3	70°	325°
JS4	50°	160°
JS5	60°	171°





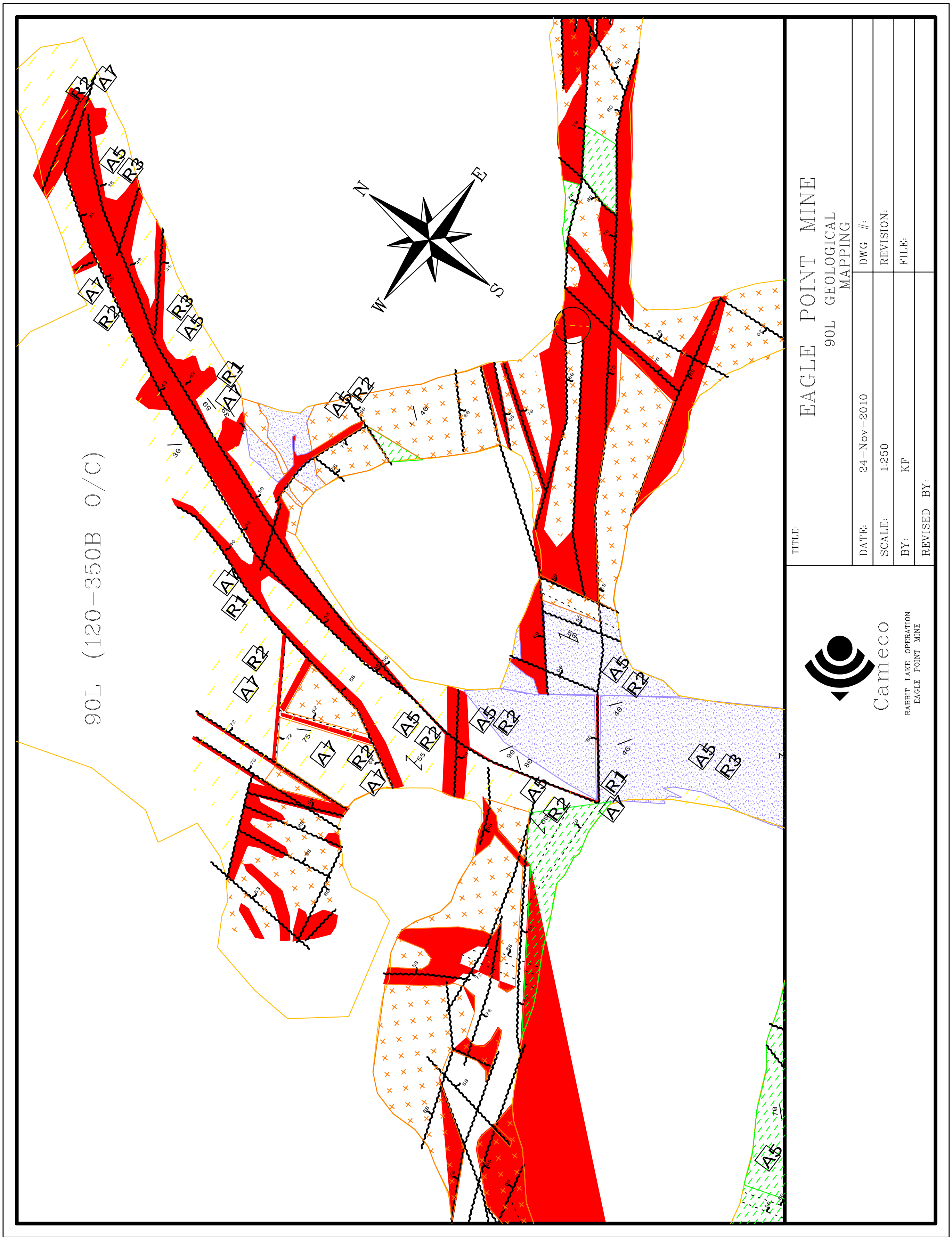
APEX 11m HT, 1822 tonnes, GRAVITY FALL

RMR FW 350B		
1) STRENGTH	25-50MPa	4
2) RQD	50%-75%	13
3) SPACING	0.3-1m-	15
4) CONDITON	TIGHT/GOUGE	12 - 6
5) GRNWTR	DRY/SLT	10 - 7
	RATING	54%-45%
STRUCTURE	FLAT	
	DESIGN (WALL)	50%

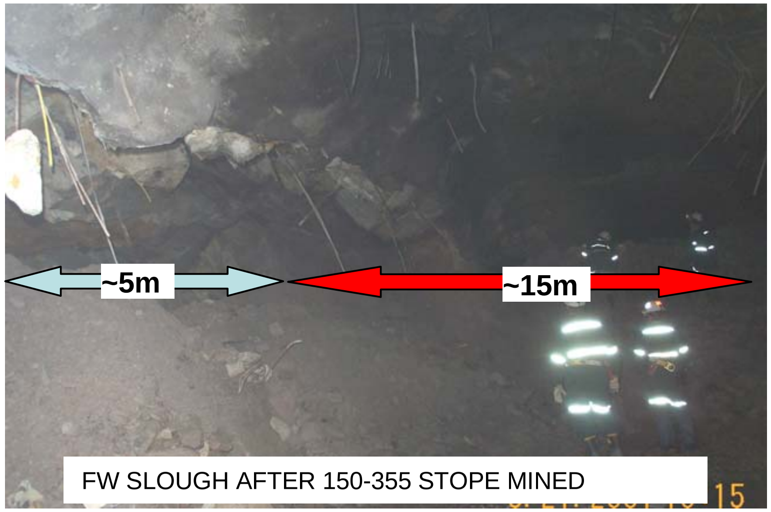


STOP #1A: 90L, 105-350B O/C

FW COLLAPSE SUBSEQUENT TO ORIGINAL 90L CAVE (2004). LARGEST SPAN APPROACHES 20m FROM HW TO FW AFTER MAY 15th COLLAPSE. AREA BACKFILLED. OBSERVED MAY 24th.



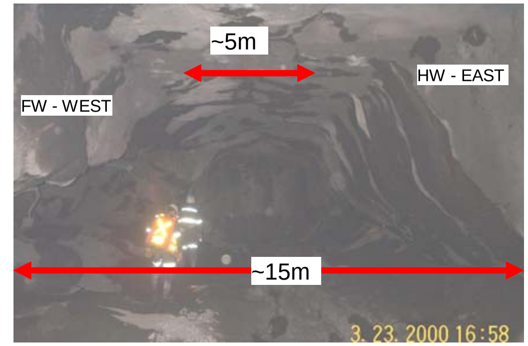
90L (120-350B O/C)
N
E
S
W
TITLE:
EAGLE POINT MINE
90L GEOLOGICAL MAPPING
DATE: 24-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE



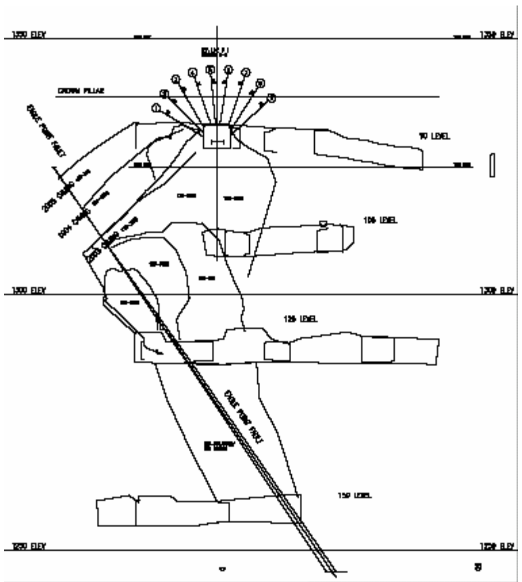
FW SLOUGH AFTER 150-355 STOPE MINED

RMR FW 350B

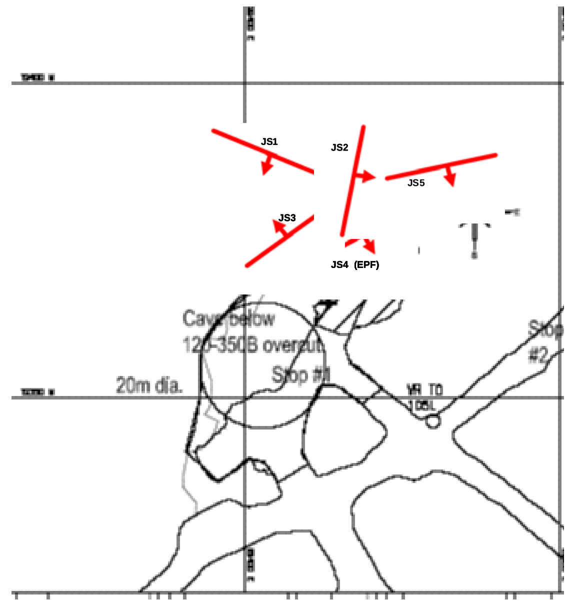
1) STRENGTH	25-50MPa	4
2) RQD	50%-75%	13
3) SPACING	0.3-1m-	15
4) CONDITON	TIGHT/GOUGE	12 - 6
5) GRNWTR	DRY/SLT	10 - 7
	RATING	54%-45%
STRUCTURE	FLAT	
	DESIGN (WALL)	50%



GEOMETRY PRIOR TO CAVE (EPM-2/04 MARCH/04 AUDIT)

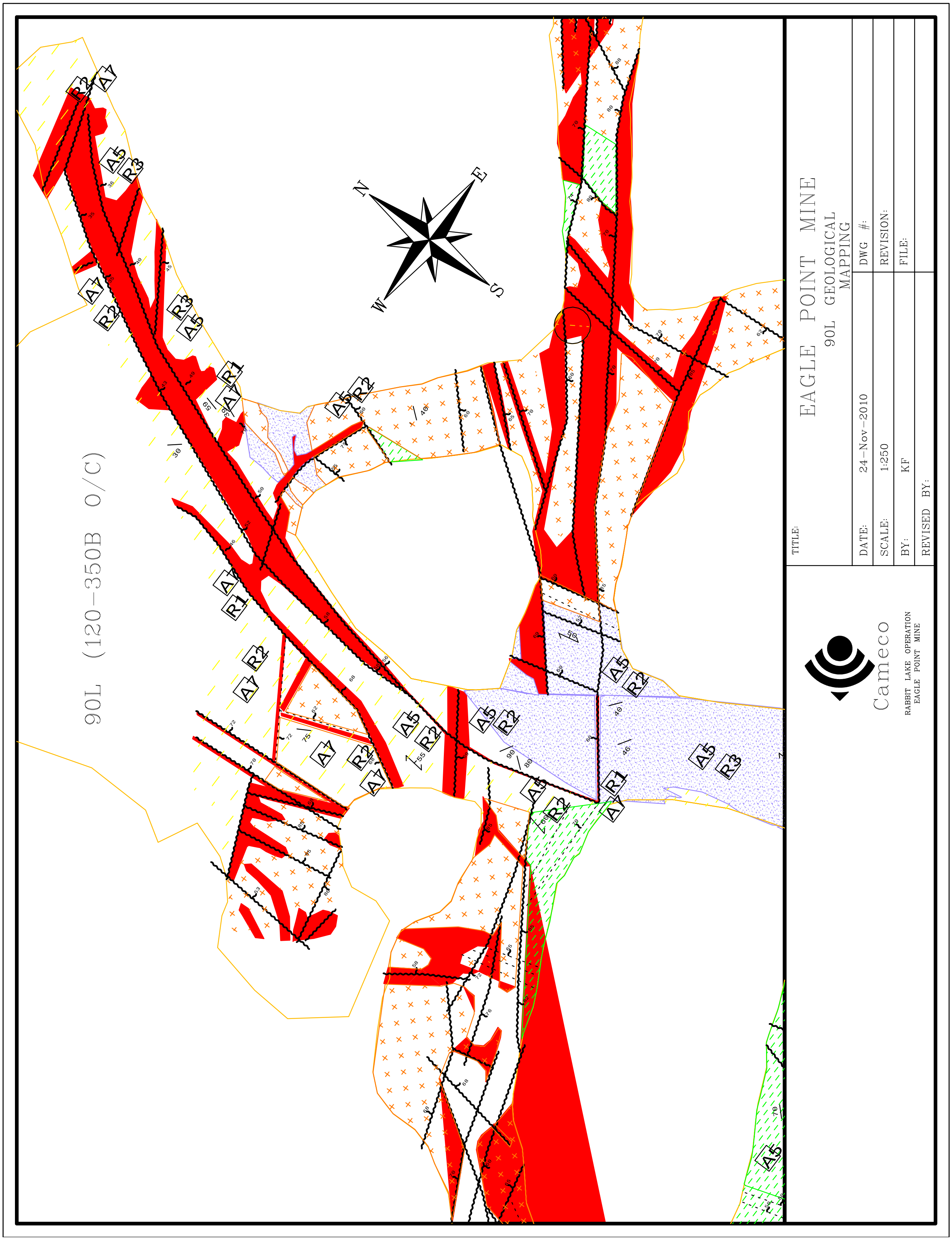


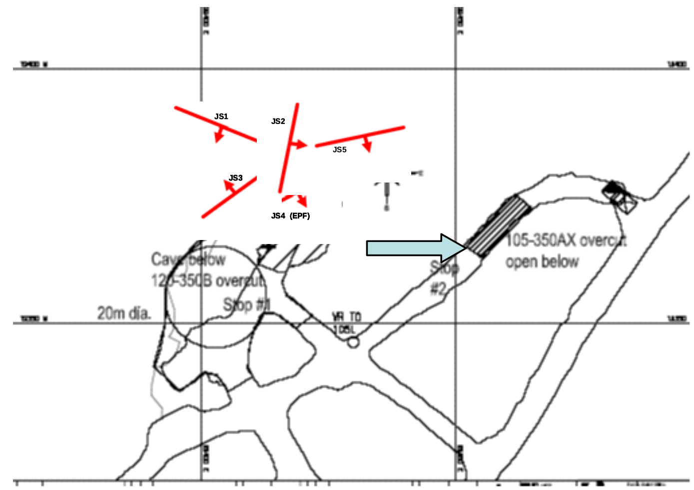
"01 CAVE" LOOKING NORTH ON 90L.120-375B STOPE CAVED FROM 120mL TO 90mL. SHOTCRETE APPROXI8MATELY 2" ON WALLS AND BACK. 2.4m LONG REBAR AND SCREEEN ON A 1.2m X 1.2m PATTERN TO WITHIN 1m OF THE SILL. 10m LONG CABLES INSTALLED FOR PERMAENT SUPPORT.



STOP #1B: 90L, 105-350B O/C

CAVE IN PROXIMITY OF 120-375B STOPE. BACK SPAN WAS INCREASED FROM 15m TO 20m WITH SUBSEQUENT MAY 15/05 FW COLLAPSE DUE TO RUN OF MUCK TO BASE OF 150-355 STOPE UPON MINING. "01 ZONE" IS STRUCTURALLY CONTROLLED BY EAGLE POINT FAULT COMBINED WITH ADVERSE STRUCTURE AS SHOWN ABOVE.

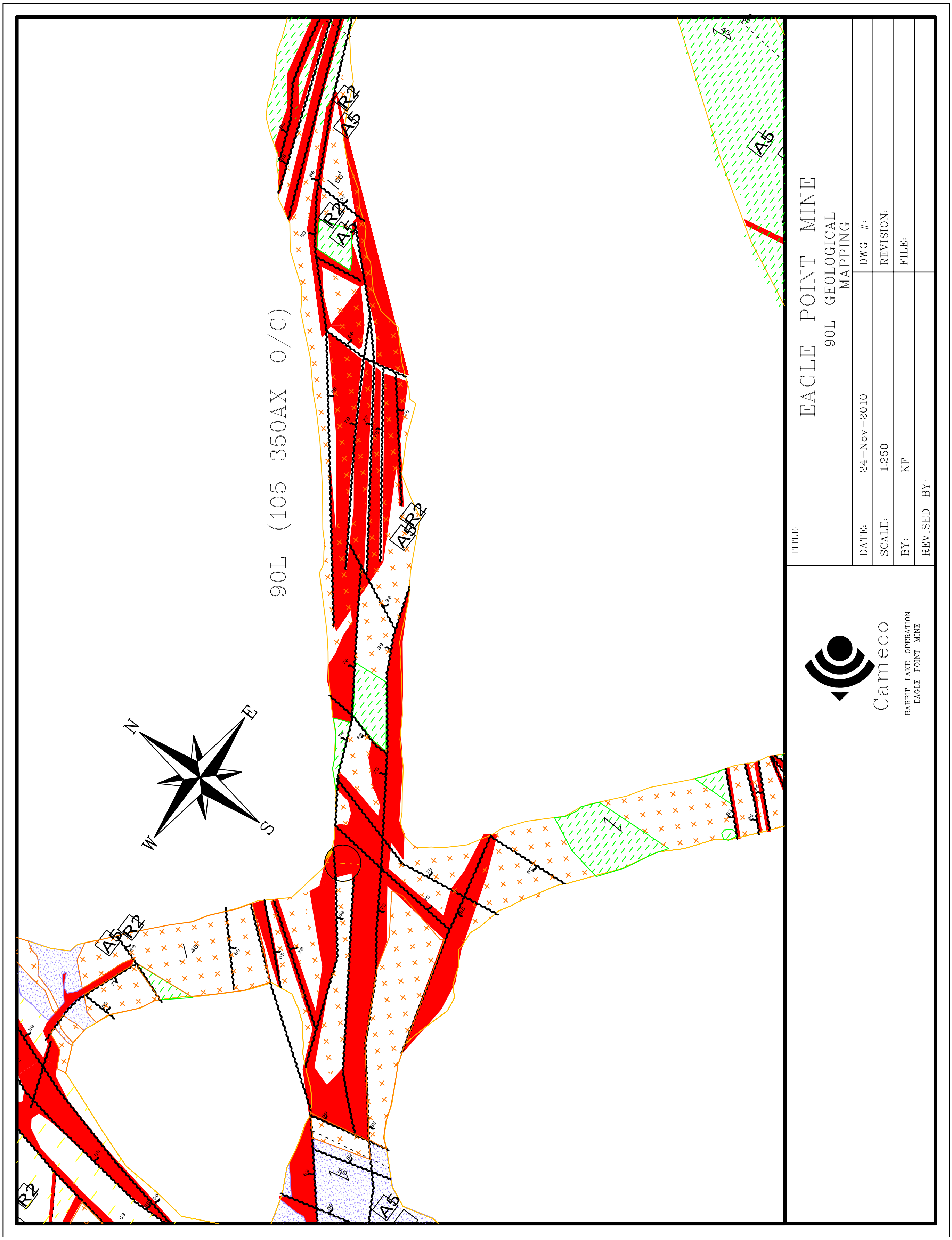

90L (120-350B O/C)
N
E
S
W
EAGLE POINT MINE
90L GEOLOGICAL
MAPPING
TITLE:
DWG #:
REVISION:
FILE:
DATE: 24-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE



RMR (HW)		
1) STRENGTH	50-100MPa	7
2) RQD	75%-90%	17
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT/SLT	12+
5) GRNWTR	DRY	10
	RATING	61%
STRUCTURE		
	DESIGN (WALL)	60-65%

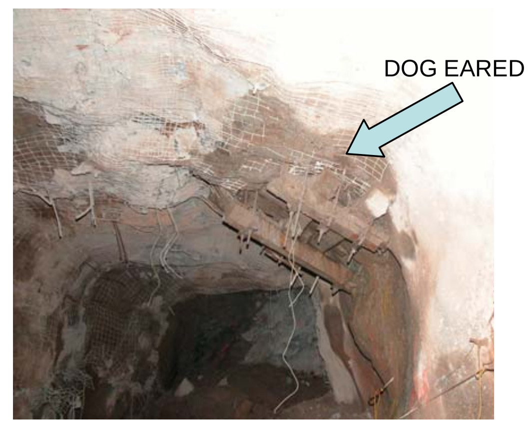
STOP #2: 90L, 105-350AX O/C STOPE

EXCELLENT STOPE. MINIMAL DILUTION. APPROXIMATE DIMENSIONS 23m HEIGHT, 20m STRIKE, 4m FW-HW. VERTICAL. ISOLATED STOPE. BACK CABLED WITH CROWN PILLAR ABOVE. DOUBLE (2X) CABLES, SHOTCRETE (2" MINIMUM) +5BOLT THROUGHT SHOTCRETE. BOLTING COMPRISED OF 8ft #6 REBAR ON 1.2m X 1.2m PATTERN (PLATED). CABLES HAVE 0.3m BULGE (NOT PLATED) AND ARE 8-10m LENGTH. STOPE STABLE. STOPE IS VERTICAL.


90L (105-350AX O/C)
N
E
S
W
A5
R2
TITLE:
EAGLE POINT MINE
90L GEOLOGICAL MAPPING
DATE: 24-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE

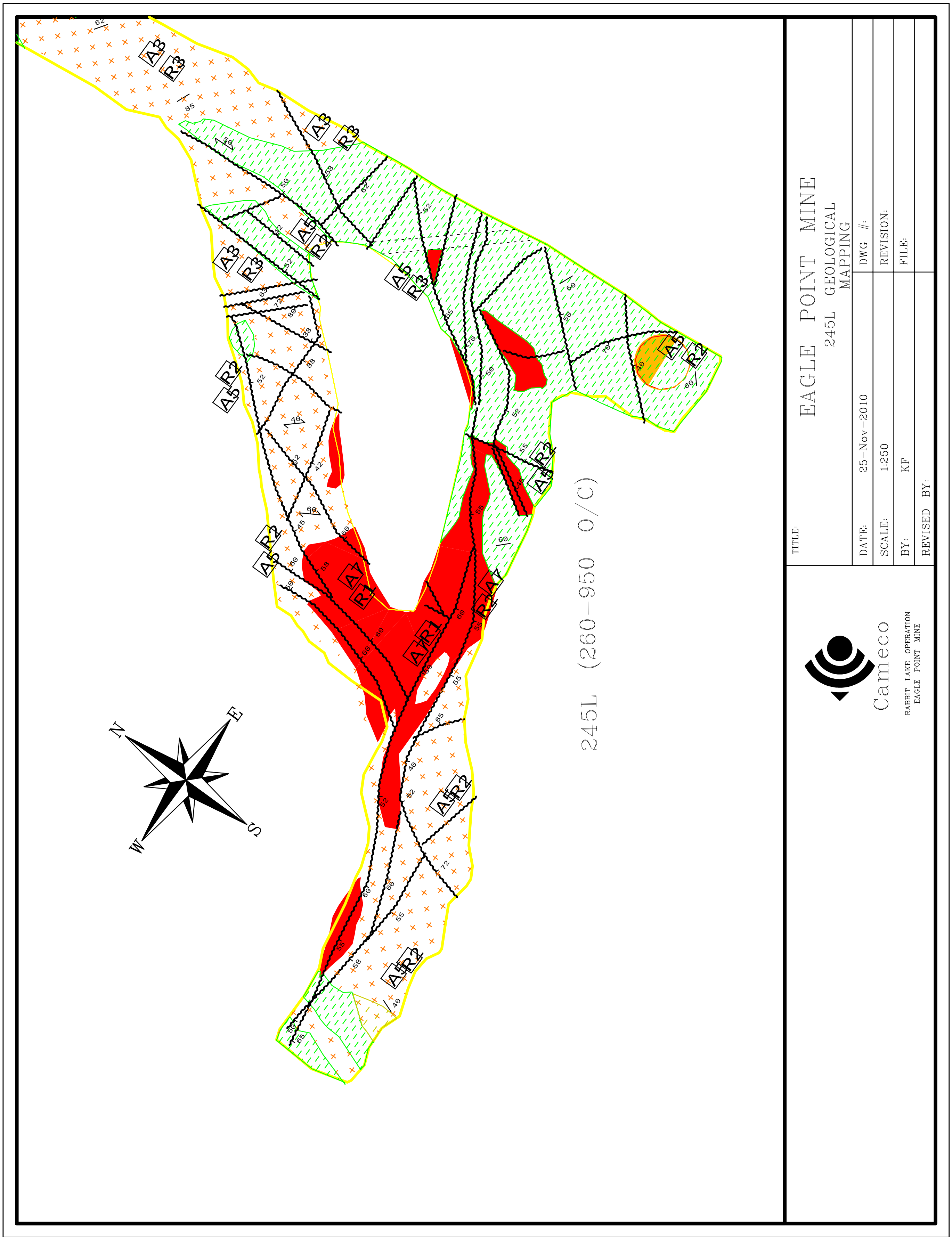
RMR (HW)

1) STRENGTH	50-100MPa	7
2) RQD	50%-75%	12
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT/SLT	12
5) GRNWTR	DRY	10
	RATING	56%
STRUCTURE		
	DESIGN (WALL)	55-60%



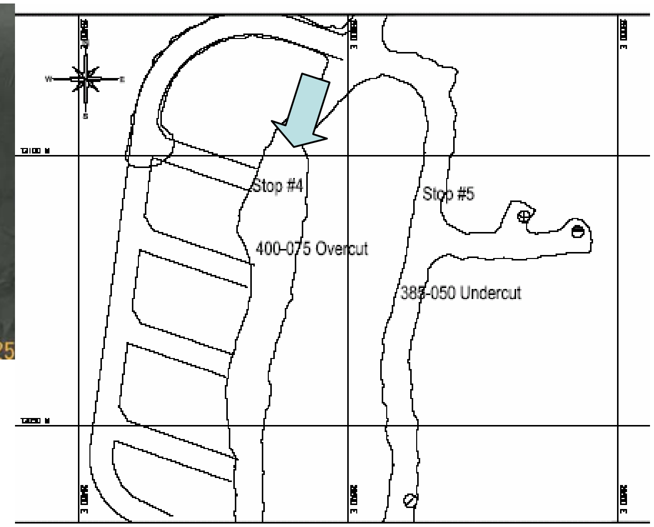
STOP #3: 245L, SUMP ZONE

245L SUMP ZONE STOPE INCLINED AT 45º APPROXIMATELY 4m FW-HW, 15m VERTICAL (245mL – 260mL), 15m STRIKE LENGTH. RAISE WITHING BACK (1.1m DIAM) "DOG EARED" FROM BACK (245mL) TO 230mL. HW HAS RMR OF 55% 6, ORE HAS RMR OF 40% (J.R.) SHOTCRETED. HW IS CABLED FROM DRILL DRIVES. APPROXIMATELY 1m ELOS.


245L (260-950 O/C)
TITLE:
EAGLE POINT MINE
245L GEOLOGICAL
MAPPING
DWG #:
REVISION:
FILE:
DATE: 25-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
N
E
S
W

RMR (HW)

1) STRENGTH	25 - 50MPa	4
2) RQD	50%-75%	13
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT/SLT	12 - 6
5) GRNWTR	DRY	10
STRUCTURE	RATING	54%-48%
	DESIGN (WALL)	50%



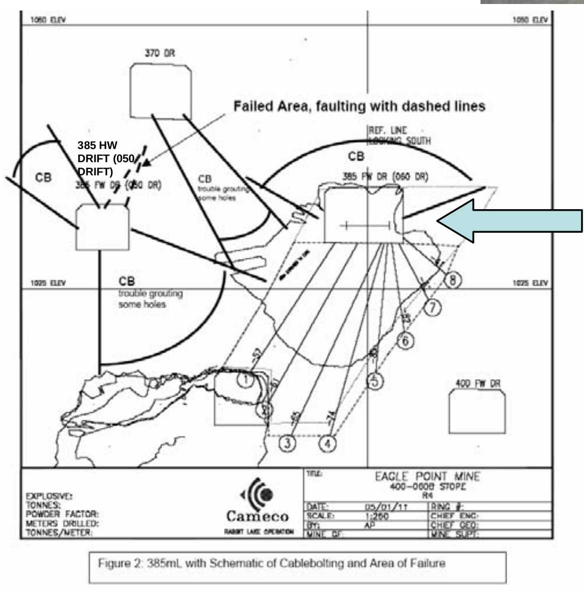


Figure 2: 385mL with Schematic of Cablebolting and Area of Failure

RMR (HW)

1) STRENGTH	25 - 50MPa-	4 - 0
2) RQD	<25%	3
3) SPACING	<50mm	5
4) CONDITON	GOUGE	6-0
5) GRNWTR	DRY	10
STRUCTURE	RATING	28%-18%
	DESIGN (WALL)	25%

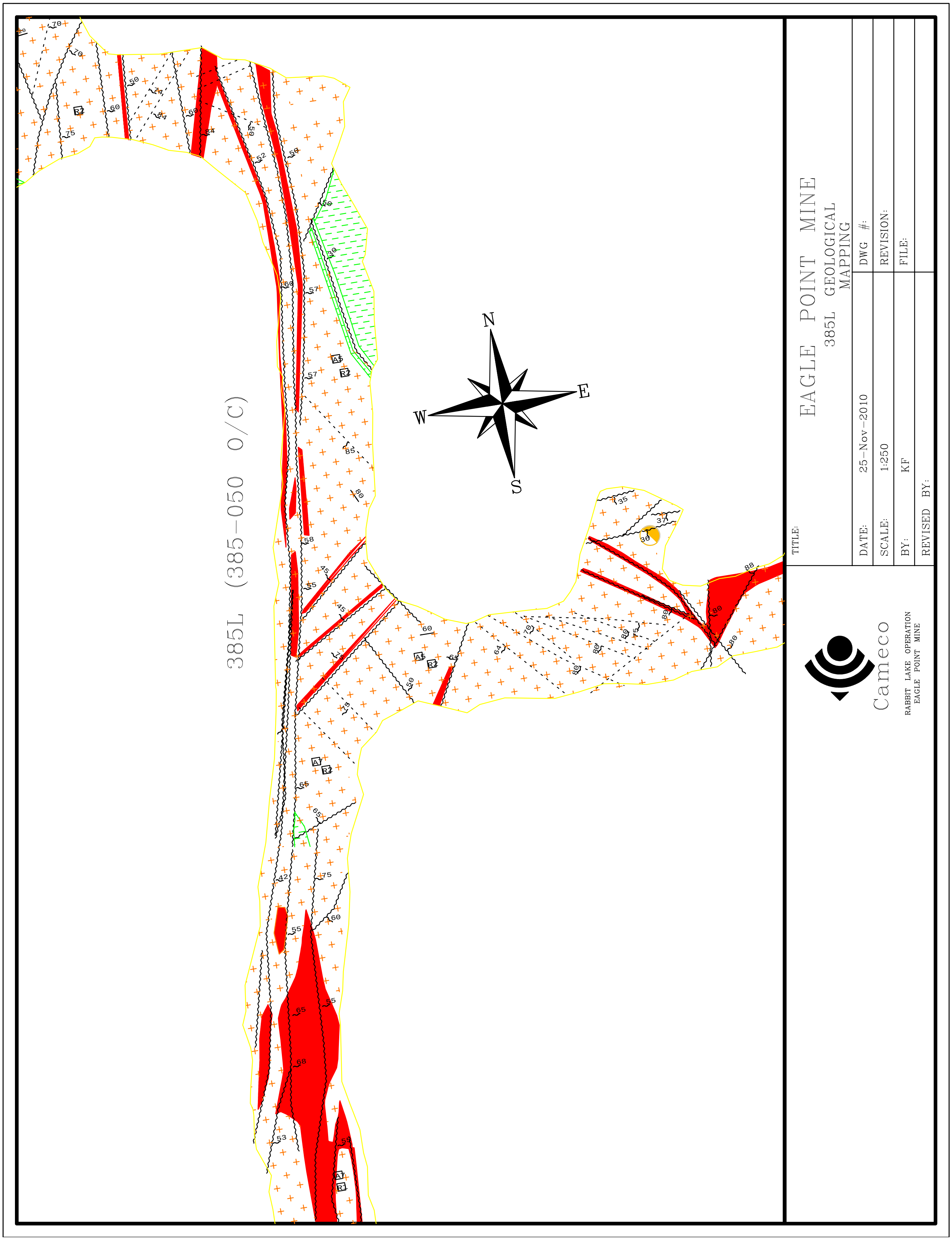
ISOLATED ZONES – WEAK – CRUMBLES
HANDLED RQD = 0%.

MECHANICAL BOLTS/REBAR/SPLITS FAILED IN TENSION – BOND HELD IN WEAK ROCK MASS WHICH WAS 25% IN ISOLATED AREAS OVERALL 50%.

STOP #4: 385L, 400-075 O/C

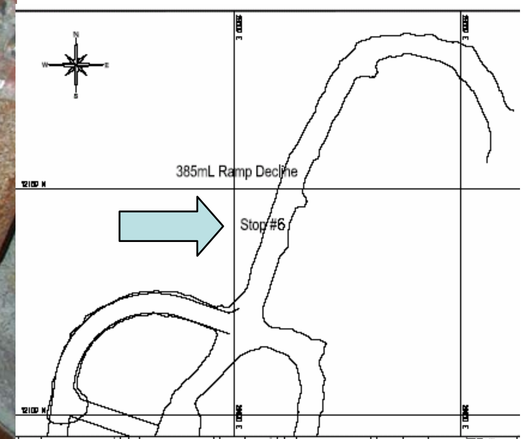
COMPLEX GEOMERY. CAVING OF HW STOPES. CAVING ABOVE 385L. REFER MEMO.

EAGLE POINT MINE
385L GEOLOGICAL MAPPING
TITLE:
DATE: 25-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
385L (400-075 O/C)
N
E
S
W


385L (385-050 O/C)
N
E
S
W
TITLE:
EAGLE POINT MINE
385L GEOLOGICAL MAPPING
DWG #:
REVISION:
FILE:
DATE: 25-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE

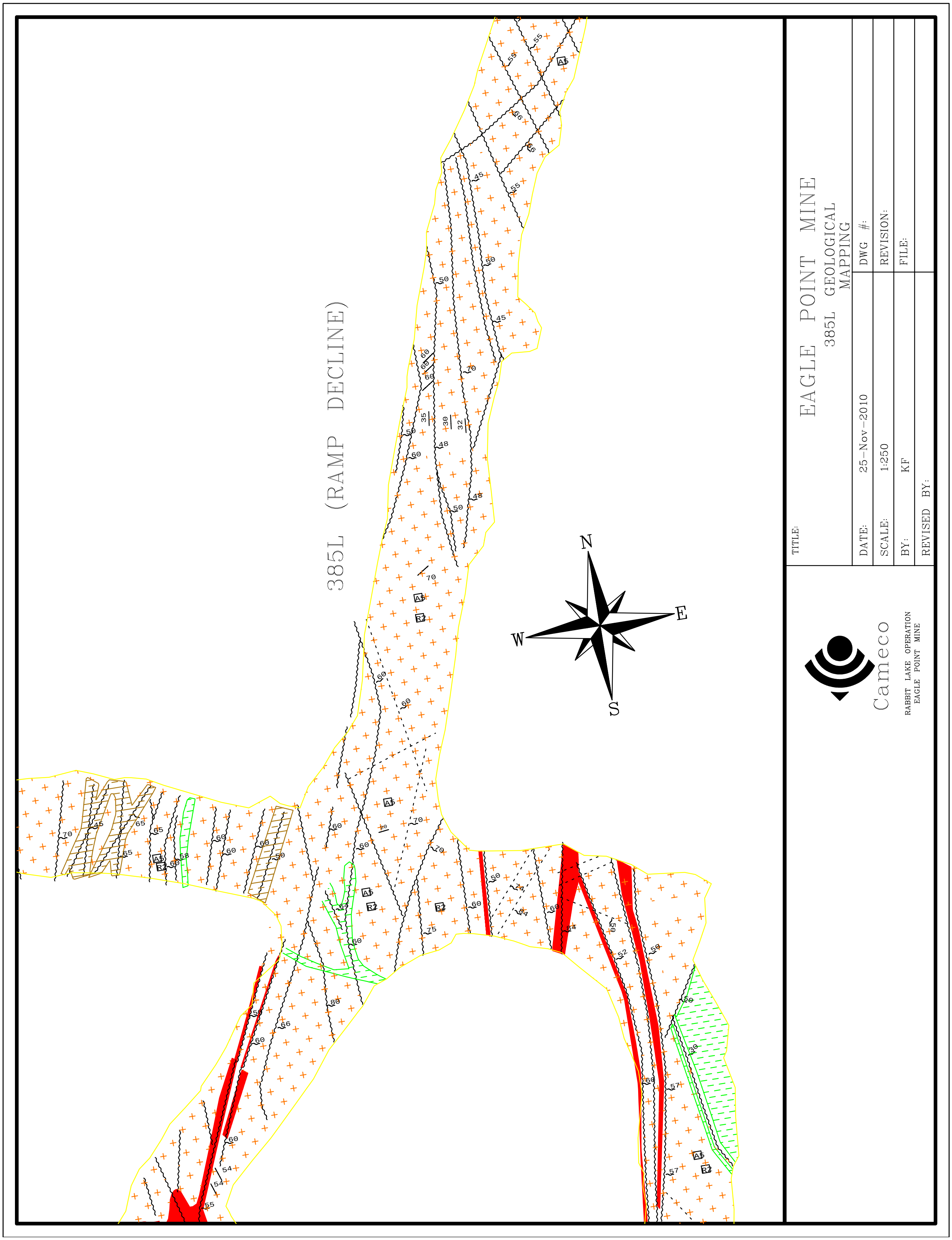
RMR (HW)

1) STRENGTH	25 - 50MPa+	7 - 4
2) RQD	25%-50%	8
3) SPACING	50-300mm	10
4) CONDITON	TIGHT/SLT	12 - 6
5) GRNWTR	DRY	10
STRUCTURE	RATING	47% - 38%
	DESIGN (WALL)	45%-40%



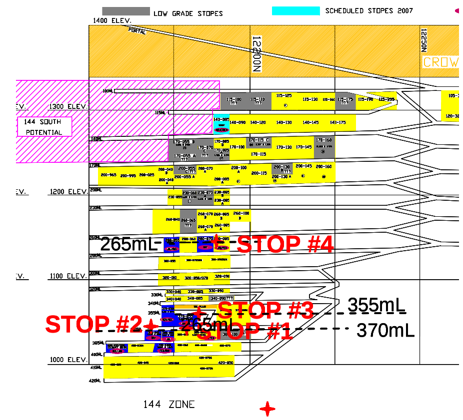
STOP #6: 385L RAMP DECLINE

RAMP DECLINE WITHIN WEAK ROCK MASS HAVING A DESIGN RMR BETWEEN 40$ - 45%.

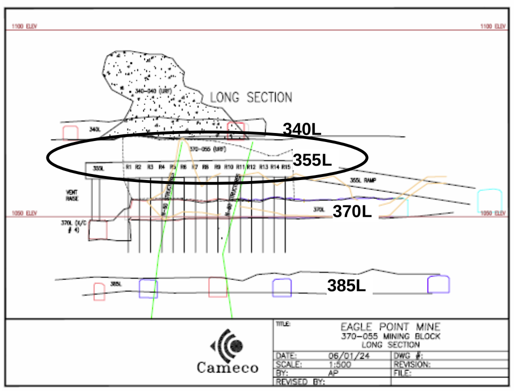

385L (RAMP DECLINE)
N
E
S
W
TITLE:
EAGLE POINT MINE
385L GEOLOGICAL MAPPING
DATE: 25-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE

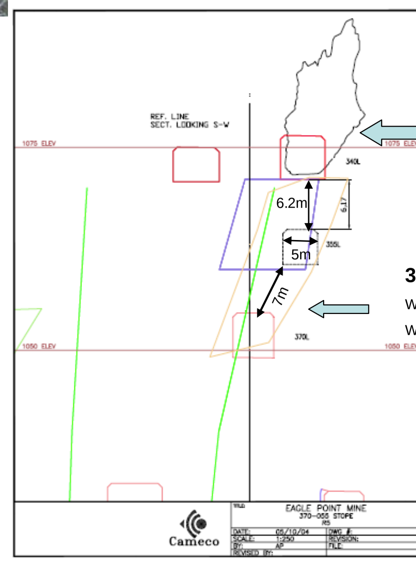
EAGLE POINT MINE LONG SECTION

144 ZONE



ANALYSIS OF 370-055 STOPE





340-355 SILL

Wp = 6.2m, Hp = 5m

Wp/Hp = 1.2

355-370 SILL

Wp = 7m, Hp = 5m

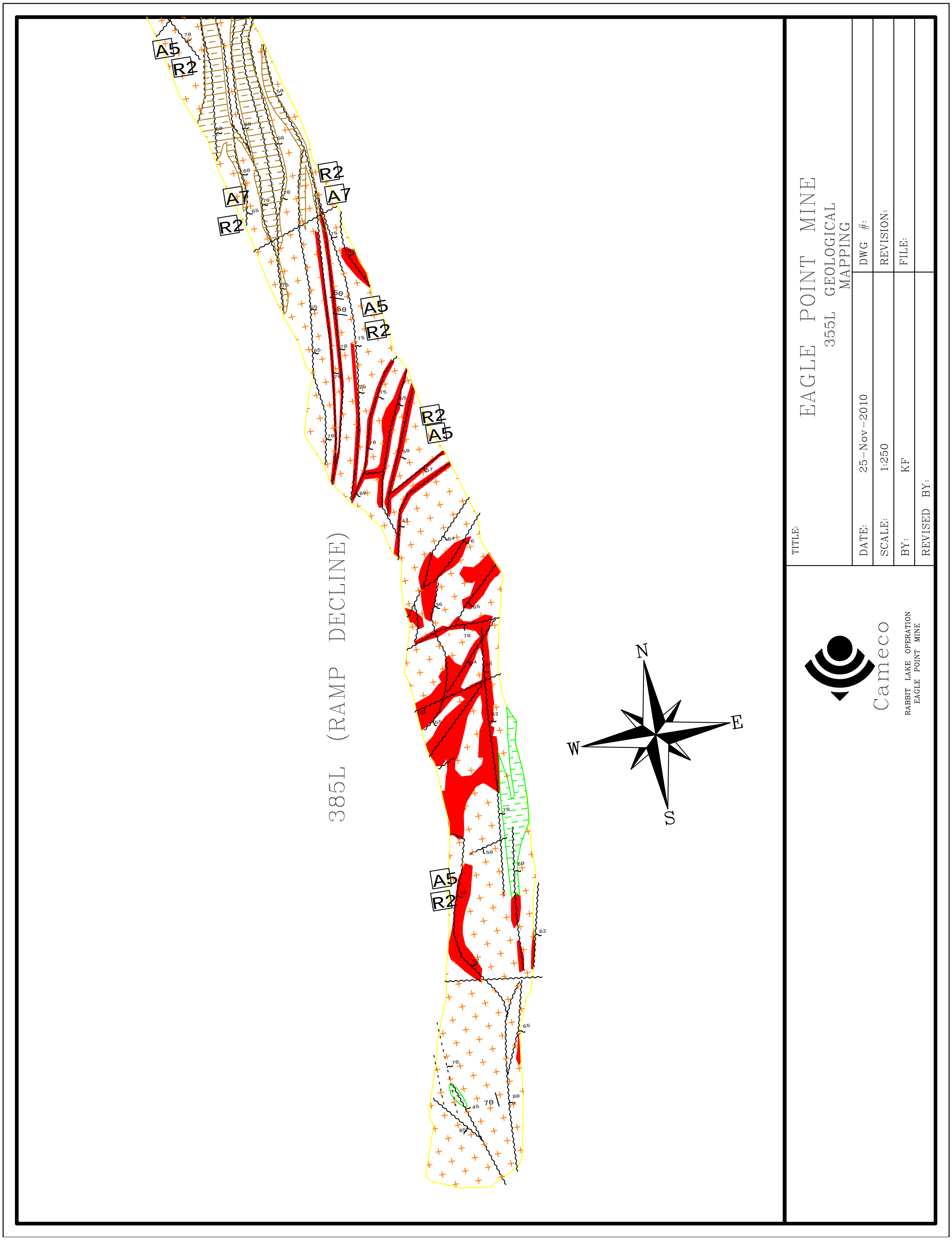
Wp/Hp=1.4

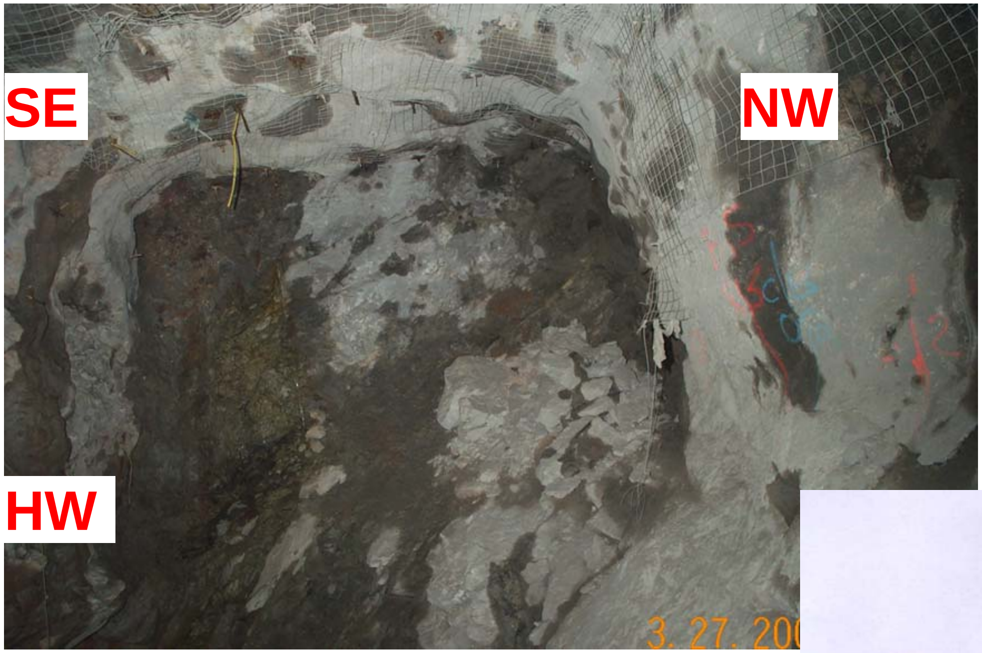
RMR BACK/ORE PEGMATITE

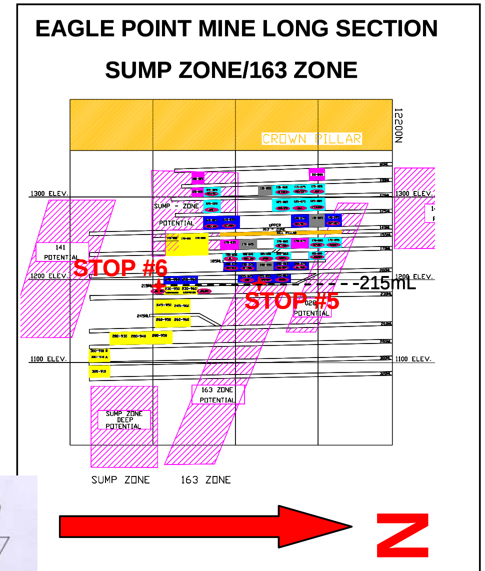
1) STRENGTH	25-50MPa	4
2) RQD	50%-75%	13
3) SPACING	300mm+	20-15
4) CONDITON	GRAPH/TIGHT	12-6
5) GRNWTR	DRY	10
	RATING	59%-48%
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	45%+

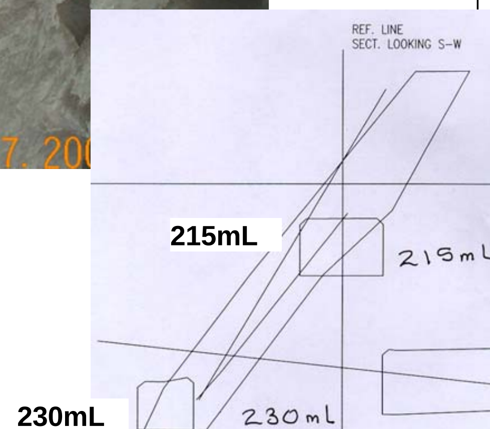
STOP #3: 355L, 370-055 ACCESS

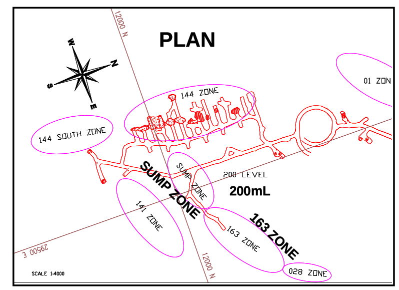
LOOKING SOUTH AT ORE FACE ON 355mL. THE FACE IS COMPRISED OF PEGMATITE HAVING AN RMR OF 45%+ . THE BACK SPAN IS 4.5m AND IS STABLE WITH SUPPORT IN PLACE (8ft REBAR + SCREEN). THE AREA WILL BE SHOTCRETED WHEN ORE EXPOSED (GAMMA). NO STRESS DETERIORATION OBSERVED AT THIS LOCATION (NO STOPE ABOVE). MOST LIKELY RMR WILL BE LOWER UPON ACCESSING FURTHER INTO ORE. DESIGN WAS 40% (FEBRUARY, 2006) WHICH WAS ESTIMATED FROM CORE LOGS (J.R.). STRUCTURE SHOULD BE ASSESSED.


EAGLE POINT MINE
355L GEOLOGICAL MAPPING
TITLE:
DATE: 25-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
385L (RAMP DECLINE)
N
E
S
W
A5
R2
A7









SECTION 230-950 STOPE R6

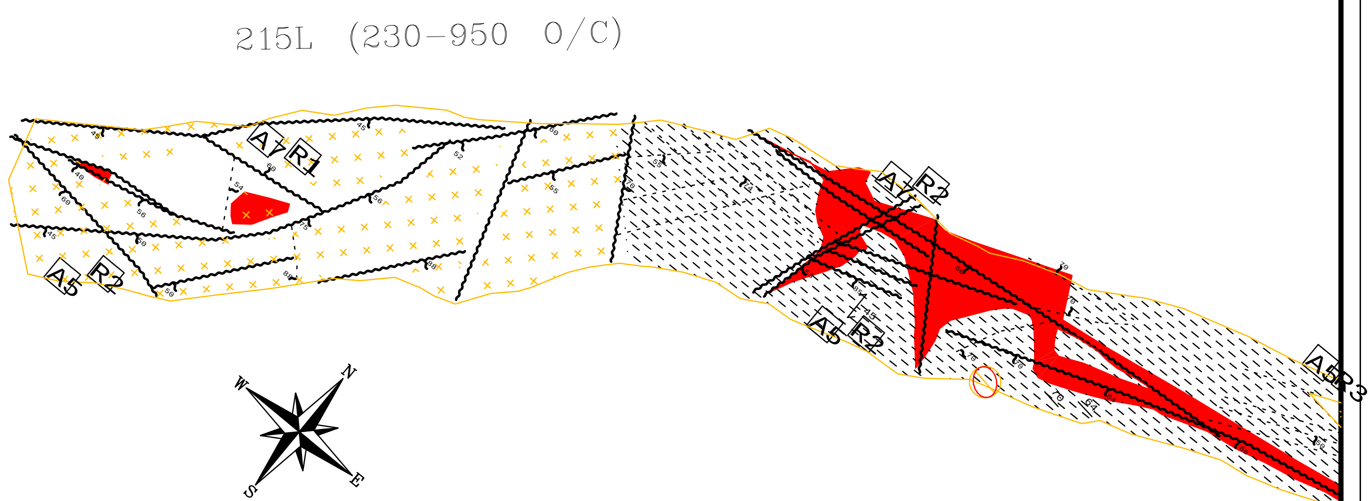
RMR HW(GNEISS)

1) STRENGTH	100-200MPa	12
2) RQD	90%	17
3) SPACING	300mm+	20-15
4) CONDITON	TIGHT-SLT	20-12
5) GRNWTR	DRY	10
STRUCTURE	RATING	79%-63%
	DESIGN	70%+

STOP #6: 215mL, 215-950 U/C (230-950 O/C) SUMP ZONE

LOOKING SOUTH WEST TOWARDS ENDWALL OF 230-950 STOPE. WALL, BACK, WALL ARE SHOTCRETED AND BOLTED THRU THE SHOTCRETE. LOOKING TOWARDS 230-950 STOPE WHICH WAS LEFT OPEN FOR VENTILATION FROM 230mL TO 215mL. THIS SEQUENCE OF STOPES WILL BE MINED OFF 215mL TO 200mL ARE AN EXTENSION OF STOPING BELOW(230mL). RMR OF HW IS 70%+ WITH RECONCILIATION (A.POP) HAVING A DESIGN ELOS OF 1.8m AND ACTUAL OF 1.5m (REALIZED). IT IS IMPORTANT TO MINIMIZE AMOUNT OF UNDERCUT OF HW SO AS TO RESTRICT PROGRESSION OF SLOUGH TO LEVELS ABOVE. HW WAS CABLED DUE TO SHALLOW DIP (50º) AND BASED UPON PAST HISTORY (SUMP ZONE). THE RMR OF THE ORE/BACK PRIOR TO SHOTCRETE WAS ESTIMATED (J.R.) AS BEING 50%-60% WITH SPAN OF 5m. THE AREA WAS STABLE WITH REBAR/SHOTCRETE/CABLE. SHOTCRETE WAS 2cm+ (1") SINCE LOW GRADE STOPE (GAMMA).

8



Cameco

RABBIT LAKE OPERATION
EAGLE POINT MINE

TITLE:	EAGLE POINT MINE 215L GEOLOGICAL MAPPING		
DATE:	25-Nov-2010	DWG #:	
SCALE:	1:250	REVISION:	
BY:	KF	FILE:	
REVISED BY:			

RMR BACK(GNEISS) DEVELOPMENT

1) STRENGTH	100-200MPa	12
2) RQD	75%-90%	17
3) SPACING	300mm+	15
4) CONDITON	TIGHT	20
5) GRNWTR	DRY	10
	RATING	74%
STRUCTURE	FLAT	-10%
	DESIGN	64%+

DEVELOPMENT IN WASTE HAS RMR BACK=74%-10%=64% (FLAT JOINTS) FOR A 4.5m SPAN. STABLE WITH 2.4m LONG MECHANICAL BOLTS. AREA IS WET. LONGTERM SUPPORT (QUALITY CONTROL) IE. AREAS WOULD BE 3yr LIFE REQUIRED.

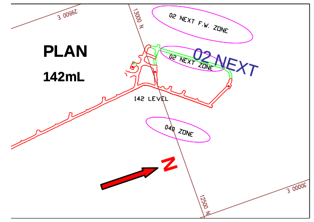
DEVELOPMENT IN ORE LOOKING TOWARDS 202-01 STOPE (NE).

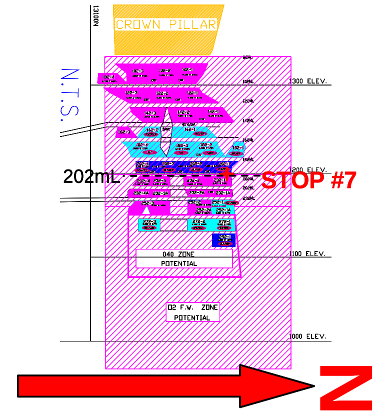
RMR HW

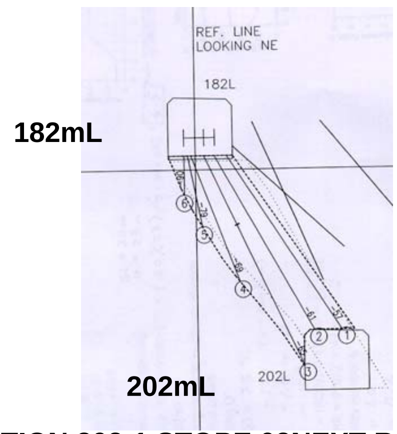
1) STRENGTH	100-200MPa	12
2) RQD	75%-90%	17
3) SPACING	50-300mm+	15-10
4) CONDITON	TIGHT-SLT	20-12
5) GRNWTR	DRY	10
	RATING	74%-61%
STRUCTURE		
	DESIGN	65%+

EAGLE POINT MINE LONG SECTION

02 NORTH EXTENSION



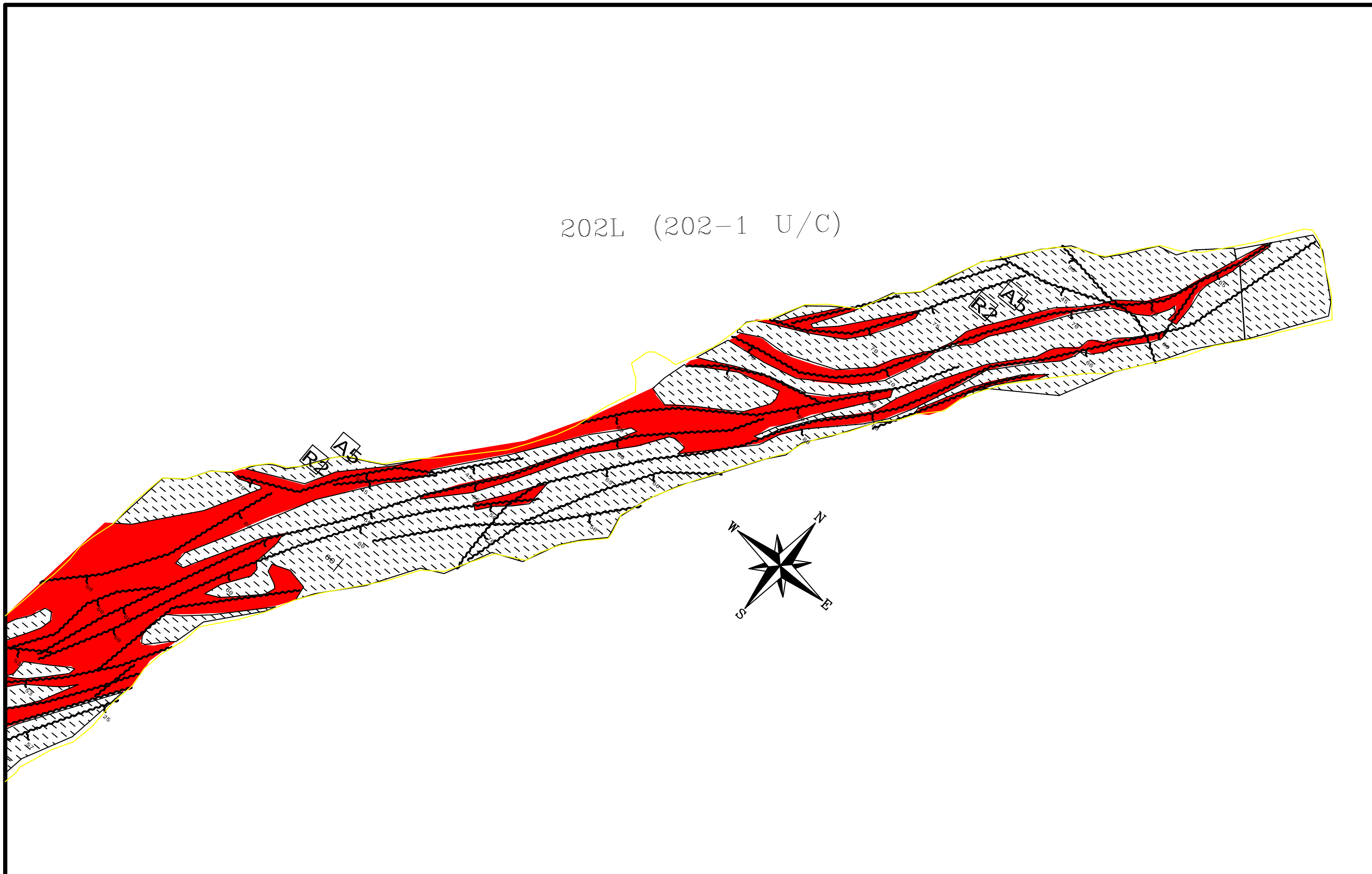




SECTION 202-1 STOPE 02NEXT R6

STOP #7: 202mL, 202-01 STOPE (02 NORTH EXTENSION)

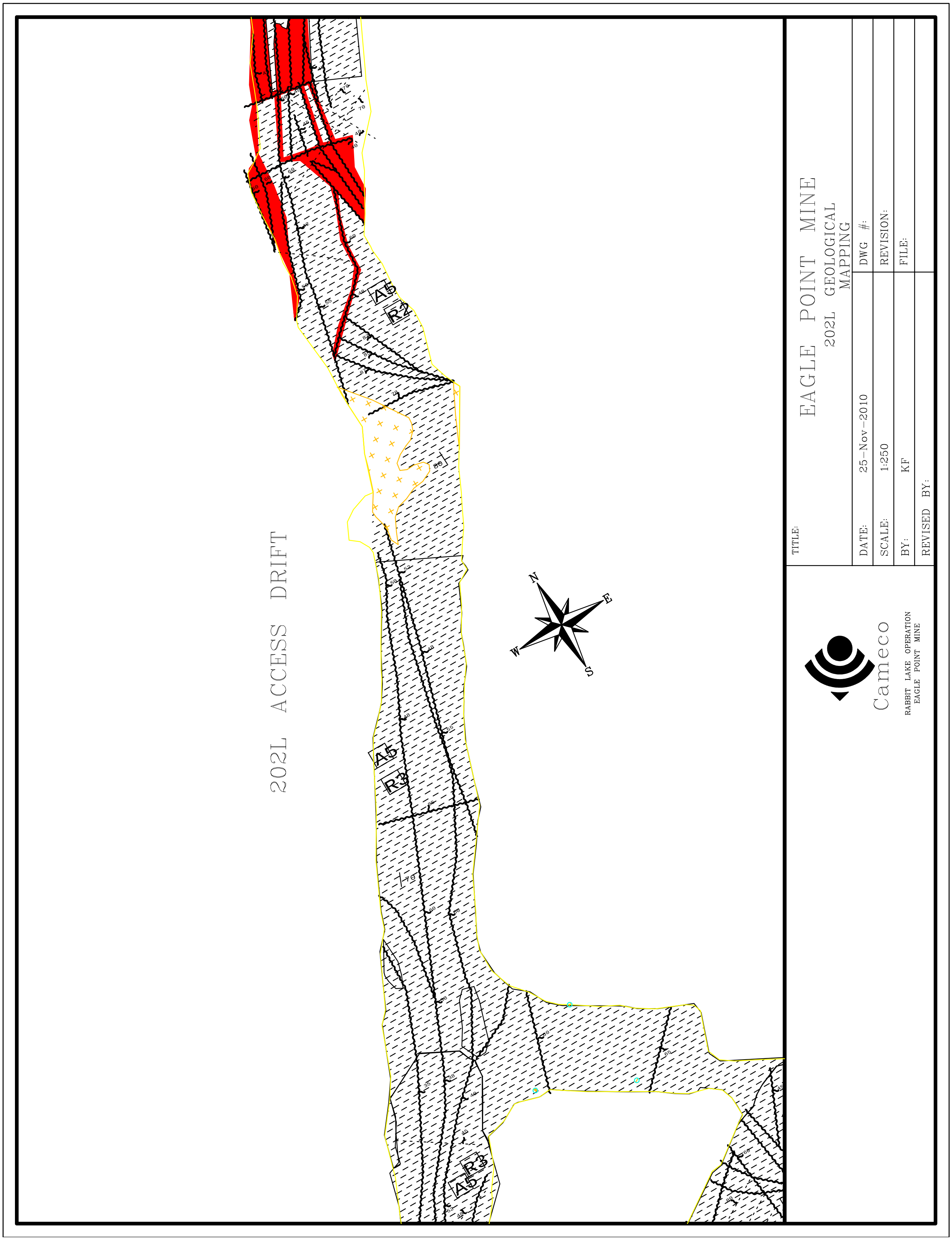
LOOKING NORTH EAST TOWARDS 202-01 STOPE. WALL, BACK, WALL ARE SHOTCRETED AND BOLTED THRU THE SHOTCRETE. THE HW OF THE 202-01 STOPE IS CABLED (~55º) WITH AN EXCELLENT HW BEING MINED HAVING A STRIKE SPAN OF 28m(L) OVER A SLOPE HEIGHT OF 25m(H) RESULTING IN A HR=6.6m. SHEET JOINTS OBSERVED IN HW WITH STOPE BROKEN TO THESE SHEET JOINTS – MINOR.NO DILUTION. OPTION OF MINING TWO(2) RINGS RESULTING IN AN INCREASED SPAN OF 6m (3m/RING) TO 34m WILL RESULT IN HR TO INCREASE TO 7.2m – STABLE WITH ONLY BLAST DAMAGE EXPECTED. THE RMR OF THE HW OBSERVED WAS 65%+. THE BACK OF THE ORE WAS ESTIMATED PRIOR TO SHOTCRETING (J.R.) TO BE 50%-60% WITH 4.5m SPAN.

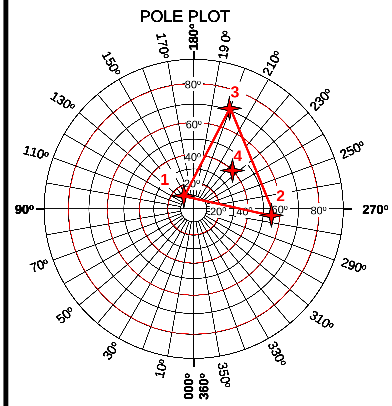


TITLE: EAGLE POINT MINE
202L GEOLOGICAL MAPPING

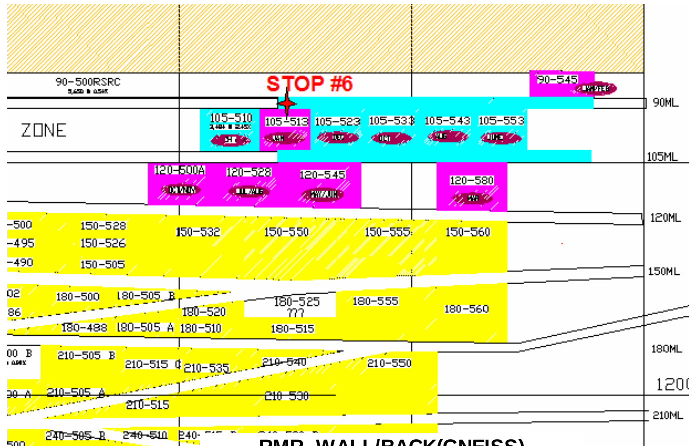
DATE:	25-Nov-2010	DWG #:	
SCALE:	1:250	REVISION:	
BY:	KF	FILE:	
REVISED BY:			

Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE


202L ACCESS DRIFT
A5
R2
A5
R3
R3
A5
N
E
S
W
TITLE:
EAGLE POINT MINE
202L GEOLOGICAL
MAPPING
DATE: 25-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE



NOTE: "SLIDING WEDGE"



RMR WALL/BACK(GNEISS)

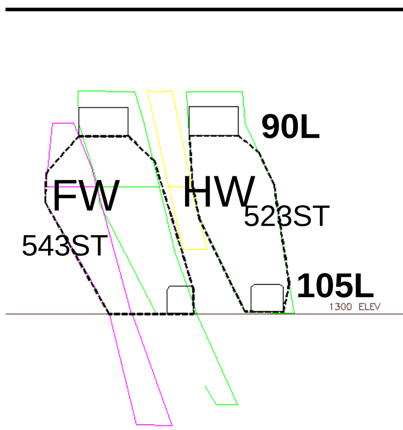
1) STRENGTH	100-200MPa	12
2) RQD	75%-50%+	13+
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT -SLT+	20-12
5) GRNWTR	DRY	10
	RATING	70%-62%
STRUCTURE	BACK	-10%
	DESIGN (BACK)	55%+

STRUCTURE MAPPED

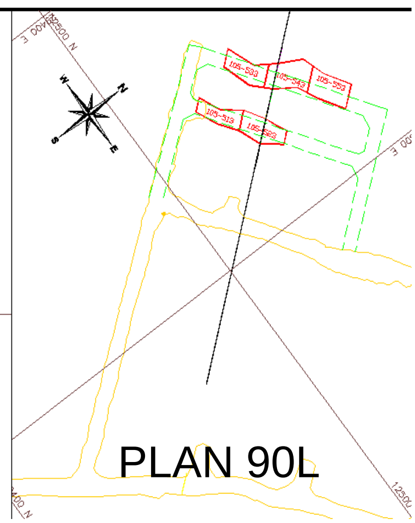
JS	DIP	DDR
1	15°	140°
2	55°	275°
3	70°	200°
4	40°	235°

TREND DEV. = 065°

03 STOPE – 90L



VERTICAL SECTION LOOKING NE



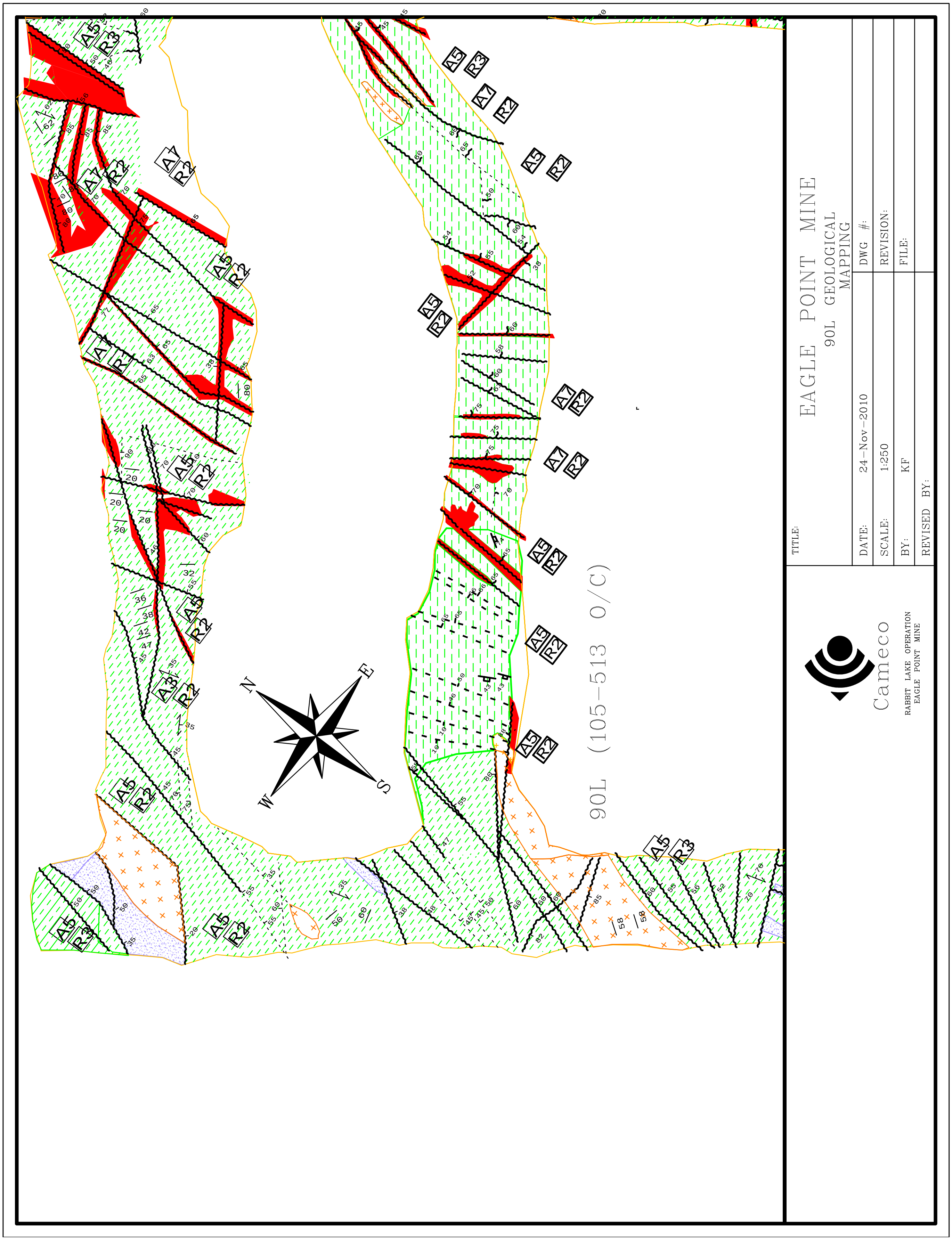
SHOULD WE MINE FW DRIFT AND HW DRIFT FIRST AND SUPPORT.

29

CAMECO CORPORATION – EAGLE POINT MINE

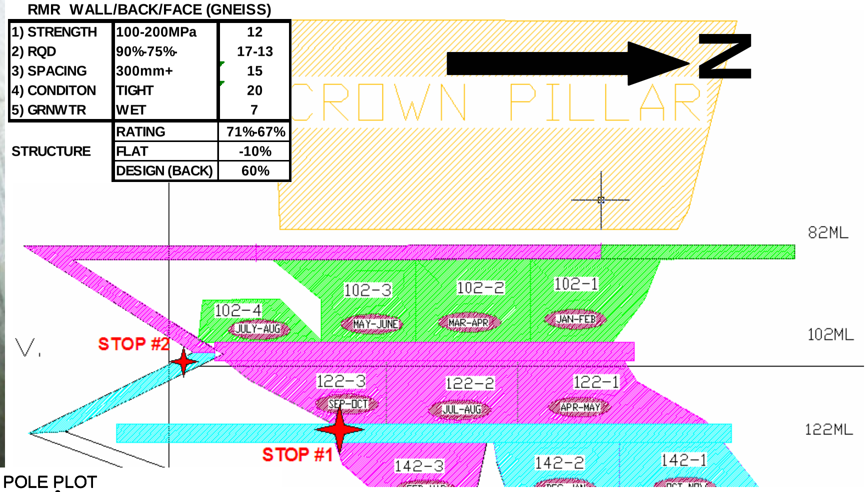
FIGURE 7: "03" STOPE ON 90L – STOP #6

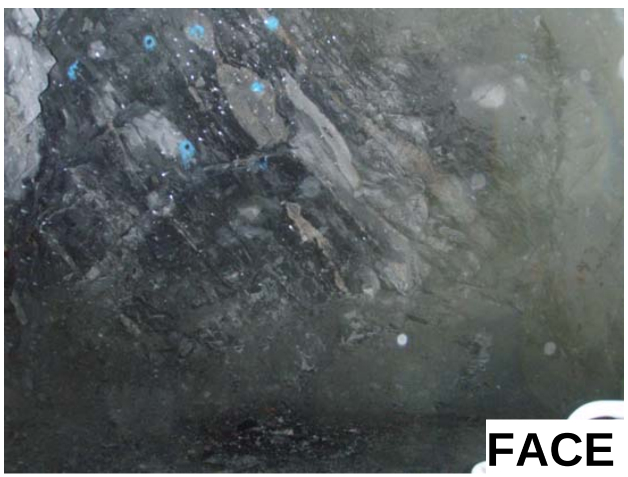
PAKALNIS & ASSOCIATES


90L (105-513 O/C)
N
E
S
W
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
TITLE:
EAGLE POINT MINE
90L GEOLOGICAL
MAPPING
DATE: 24-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:

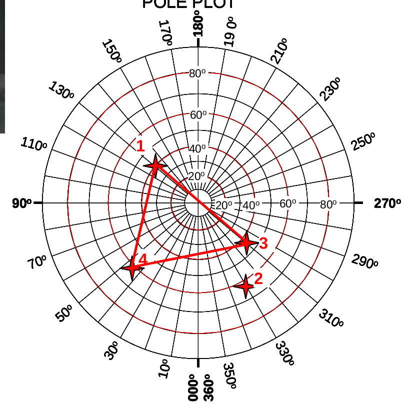
RMR WALL/BACK/FACE (GNEISS)

1) STRENGTH	100-200MPa	12
2) RQD	90%-75%	17-13
3) SPACING	300mm+	15
4) CONDITON	TIGHT	20
5) GRNWTR	WET	7
	RATING	71%-67%
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	60%





POLE PLOT



STRUCTURE MAPPED

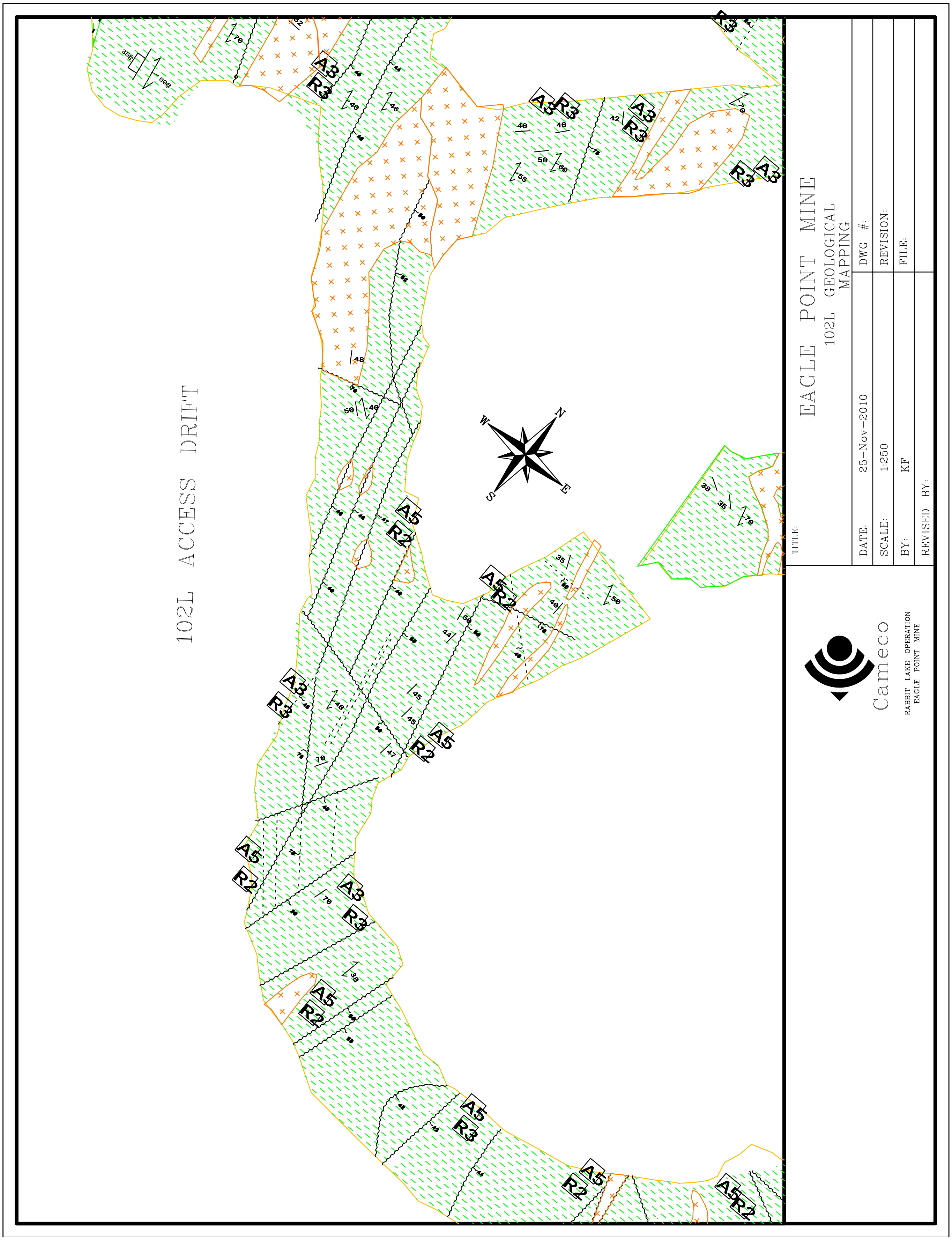
JS	DIP	DDR
1	40°	130°
2	65°	330°
3	45°	310°
4	60°	104°

TREND DEV. = 050°

102 LEVEL PLAN

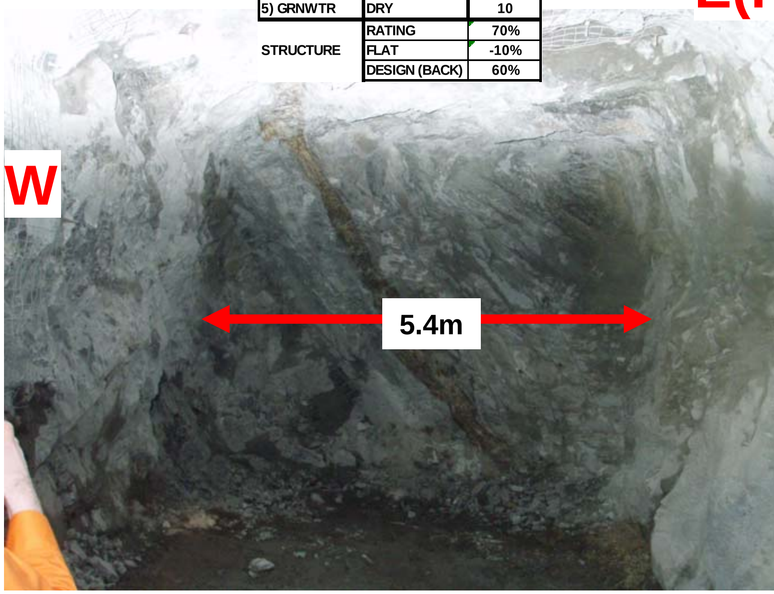
STOP #2: 102 RAMP TOWARDS 122-1/2/3 O/C OR 102-1/2/3/4 U/C

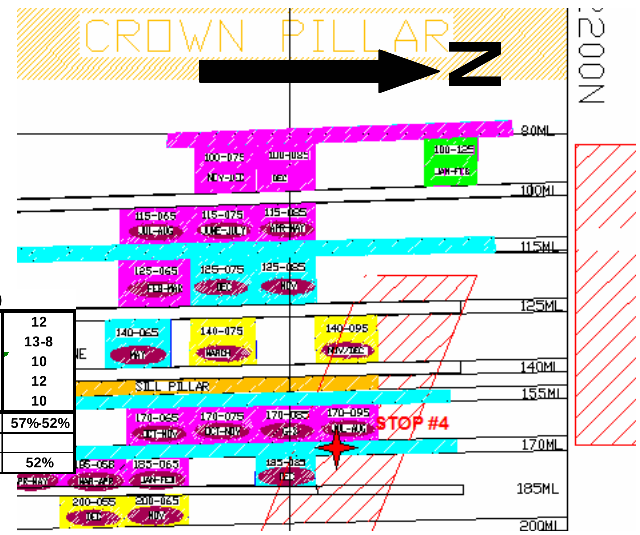
LOOKING NORTH EAST. SPAN IS 6.5m-7m PERMANENT RAMP WILL BE INCREASED TO 9m (PLAN). BLOCKY GROUND RECOGNIZED BY OPERATIONS/ENGINEERING STD BOLTING AUGMENTED WITH EXTENSION DYWIDAG 4.8m LONG (#7) ON 1.5m X 1.5m PATTERN ~ 3 BOLTS ACROSS SPAN. NOTE BLOCKY GROUND – POTENTIAL FOR "DEAD WEIGHT" WEDGE.


102L ACCESS DRIFT
TITLE:
EAGLE POINT MINE
102L GEOLOGICAL MAPPING
DATE: 25-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE

RMR BACK/HW (GNEISS)

1) STRENGTH	100-200MPa	12
2) RQD	75%-50%	13
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT	20
5) GRNWTR	DRY	10
STRUCTURE	RATING	70%
	FLAT	-10%
	DESIGN (BACK)	60%





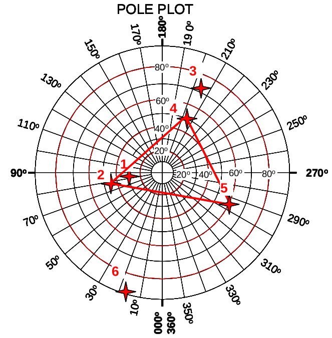
RMR HW (GNEISS)

1) STRENGTH	100-200MPa	12
2) RQD	75%-50%	13-8
3) SPACING	50-300mm	10
4) CONDITON	SLT+	12
5) GRNWTR	DRY	10
STRUCTURE	RATING	57%-52%
	PARALLEL	
	DESIGN (HW)	52%

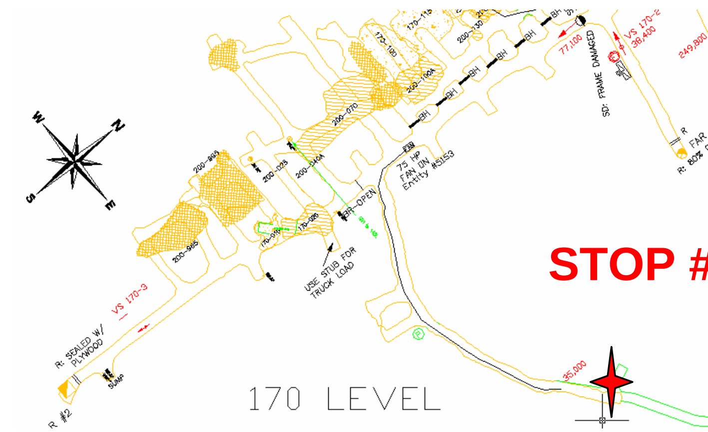
STRUCTURE MAPPED

JS	DIP	DDR
1	30°	080°
2	45°	077°
3	75°	205°
4	50°	205°
5	60°	295°
6	90°	015°

TREND DEV. = 065°



NOTE: "DEAD WEIGHT WEDGE"

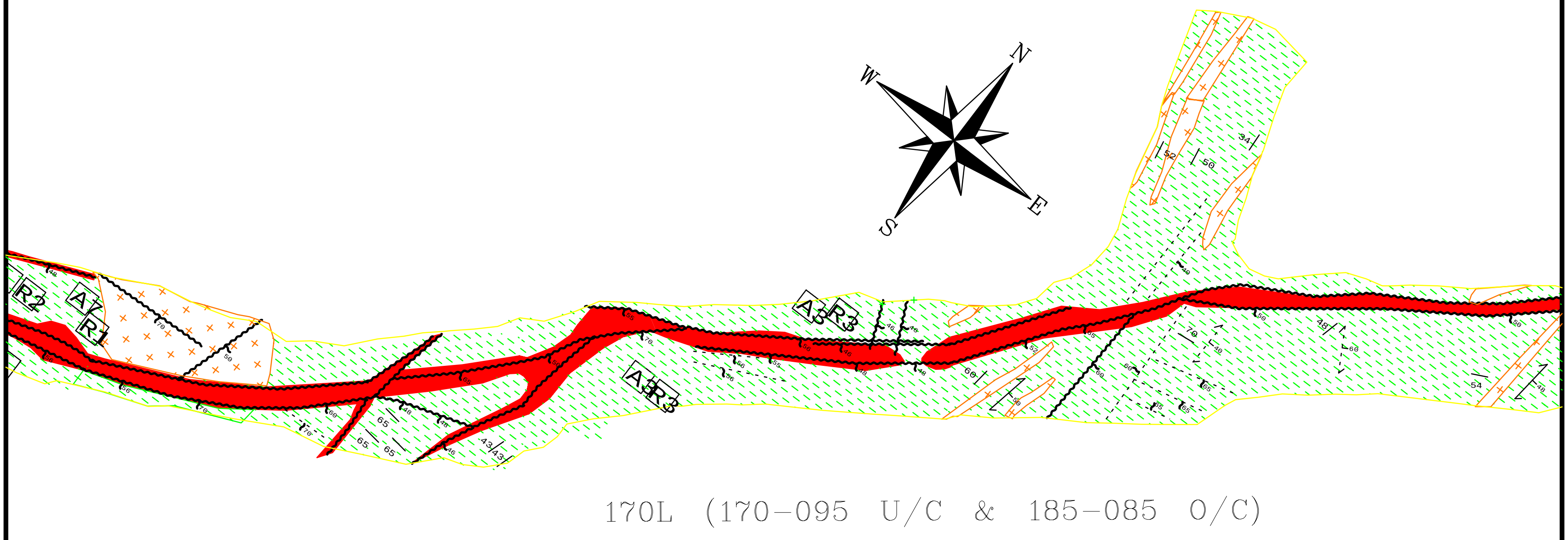


170 LEVEL PLAN

STOP #4: 170L, 170-095 U/C, 185-085 O/C.

LOOKING NORTH EAST. HW RMR IS 52% (Q' = 2.4) WITH PARALLEL JOINTS TO HW (B=0.2), DIP OF STOPE IS 60º RESULTING IN A N=Q X A X B X C = 2.4 X 1 X 0.2 X 5 = 2.4. HYD. RADIUS PLANNED IF 5-6m+ WHICH WOULD INDICATE ELOS OF 2m+. BACK/FW HAS RMR OF 70% WITH BACK DUE TO FLAT JOINTS OF 60% WITH 5.4m SPAN WHICH IS STABLE (RMR). POTENTIAL "DEAD WEIGHT" FAILURE DUE TO STRUCTURLE, HOWEVER CONFINED BY 2.4m LONG DYWIDAG (#7) BOLTS ON A 1.2m X 1.2m PATTERN. NOTE IN ADDITION WOULD HAVE SHOTCRETE.. NO DESIGN YET AS NOT EXPOSED

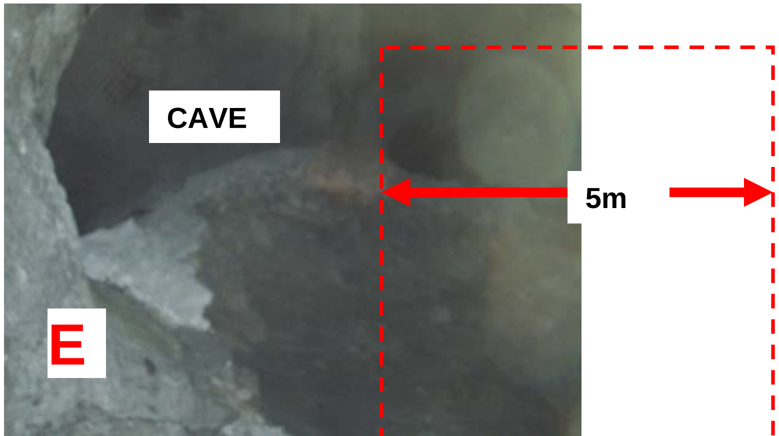
16



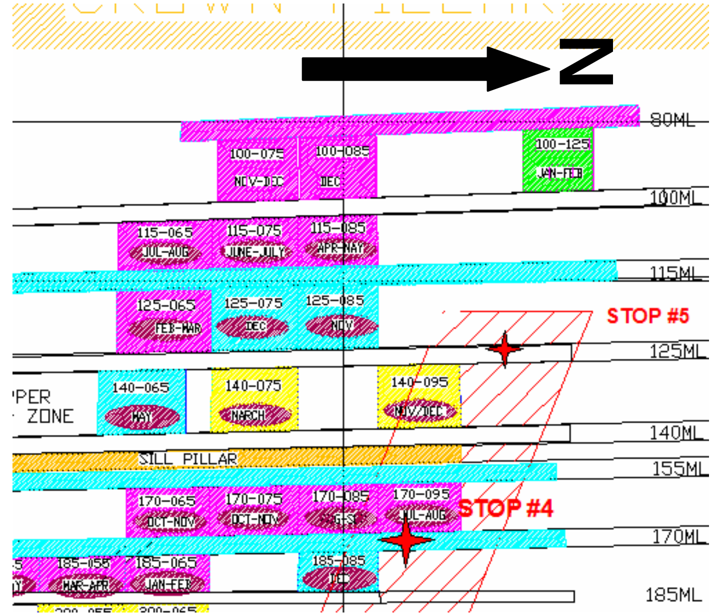
Cameco

RABBIT LAKE OPERATION
EAGLE POINT MINE

TITLE: EAGLE POINT MINE 170L GEOLOGICAL MAPPING	
DATE: 25–Nov–2010	DWG #:
SCALE: 1:250	REVISION:
BY: KF	FILE:
REVISED BY:	

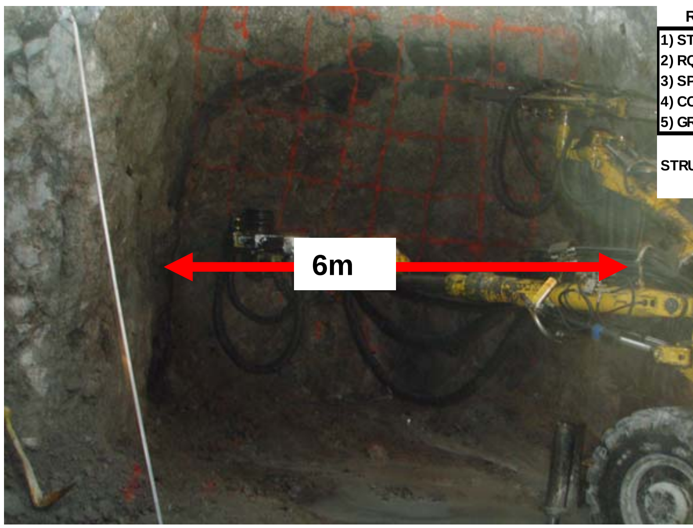


STOP 5A: LOOKING STH DOWN EXPLORATION DRIVE IN PROXIMITY OF CAVE ON EAST WALL(LEFT). CAVE MIGRATION FROM STOPE BELOW WHICH WAS INTERCEPTED AS 125L EXPLORATION DRIVE WAS BEING DRIVEN.

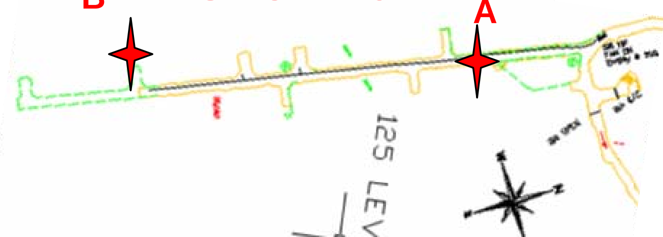


RMR WALL/BACK(PEGMATOID)

1) STRENGTH	100-200MPa	12
2) RQD	75%-50%+	13
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT -SLT+	20-12
5) GRNWTR	DRY	10
STRUCTURE	RATING	70%-62%
	BACK	-10%
	DESIGN (BACK)	55%+



“HALF BARRELS” IN BACK

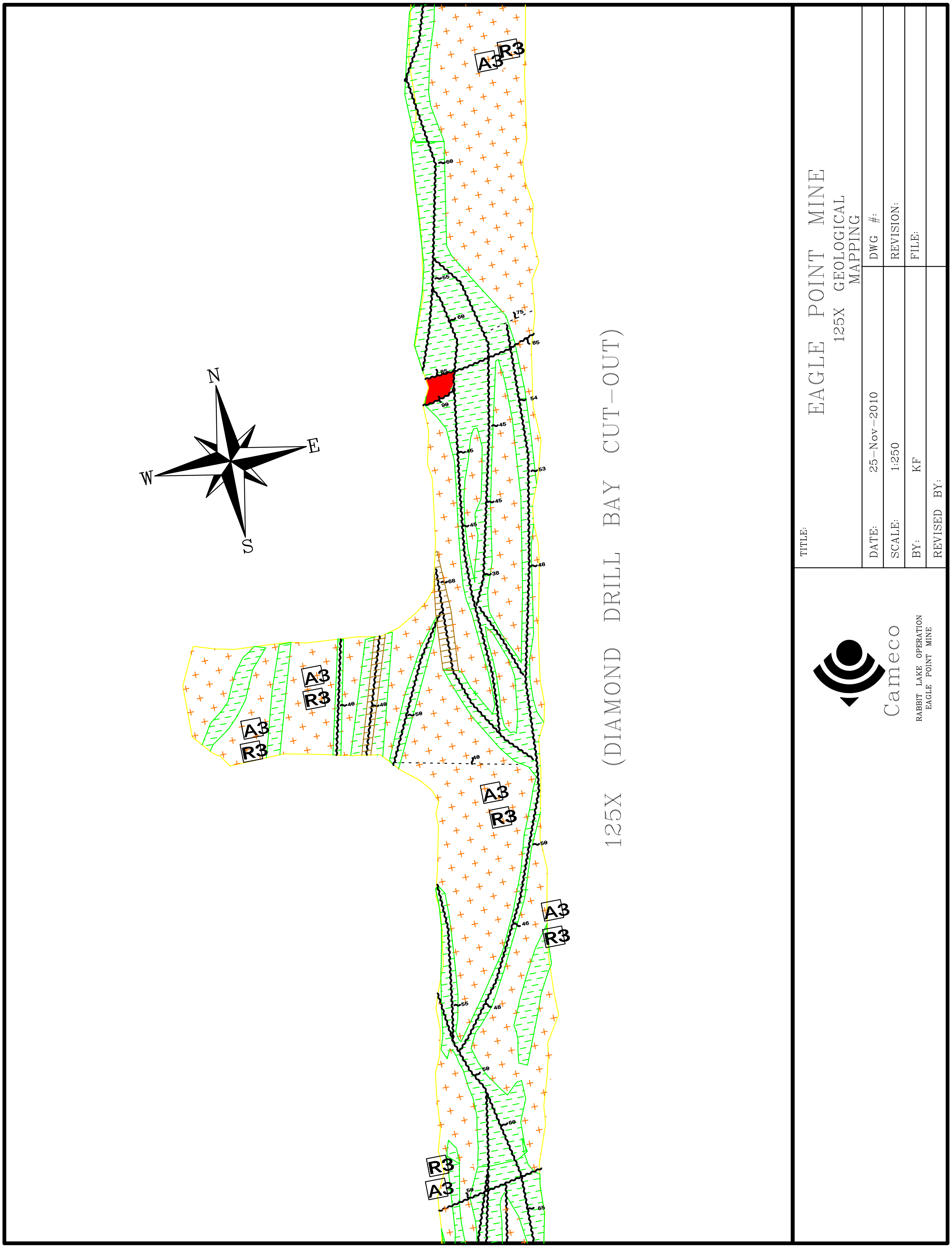


125 LEVEL PLAN

STOP 5B: LOOKING WEST AT DIAMOND DRILL BAY CUTOUT. RMR IN EXCESS OF 65% IN WALL WITH 55%+ DUE TO FLAT JOINTS IN BACK. DIMENSIONS OF OPENING ARE 6m X 6m. “HALF BARRELS” OBSERVED IN BACK.

STOP #5: 125 EXPLORATION DRIFT (144 SOUTH EXPLORATION)

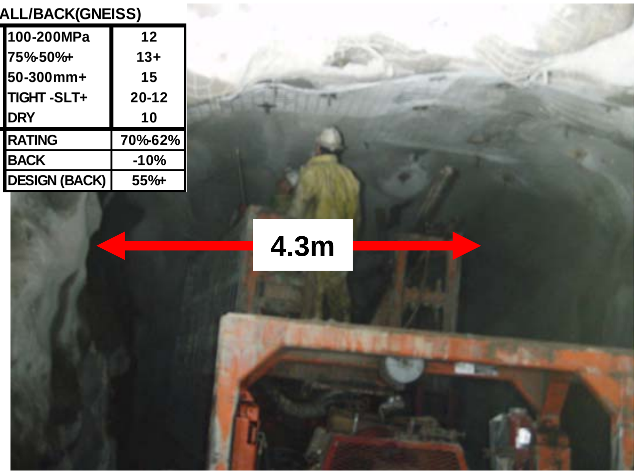
EXPLORATION DRIVE THROUGH CAVE INTERCEPTED UPON DRIVING (STOP #5A) AND DIAMOND DRILL BAY – STOP #5B.


EAGLE POINT MINE
125X GEOLOGICAL MAPPING
TITLE:
DWG #:
REVISION:
FILE:
DATE: 25-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
125X (DIAMOND DRILL BAY CUT-OUT)
N
E
S
W
A3
R3

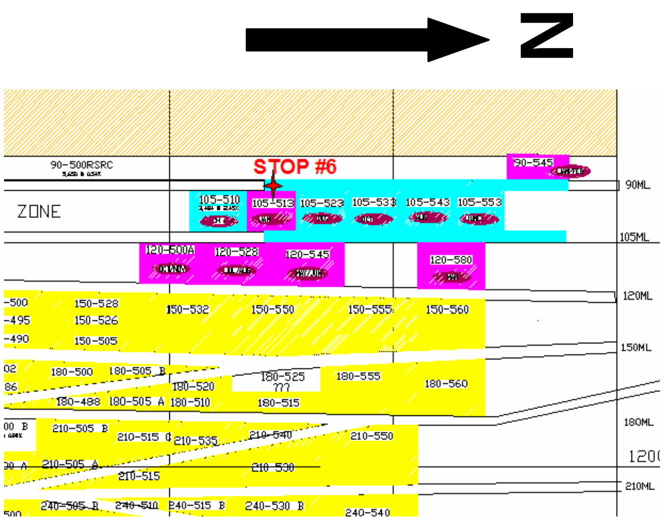
RMR WALL/BACK(GNEISS)

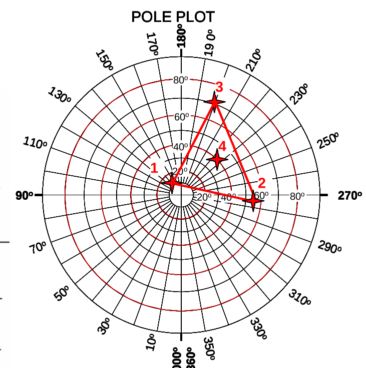
1) STRENGTH	100-200MPa	12
2) RQD	75%-50%+	13+
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT -SLT+	20-12
5) GRNWTR	DRY	10
	RATING	70%-62%
STRUCTURE	BACK	-10%
	DESIGN (BACK)	55%+

NORTHWEST WALL(FW)



SOUTHEAST WALL(HW) – PARALLEL JOINTS TO WALL





NOTE: "SLIDING WEDGE"

STRUCTURE MAPPED

JS	DIP	DDR
1	15°	140°
2	55°	275°
3	70°	200°
4	40°	235°

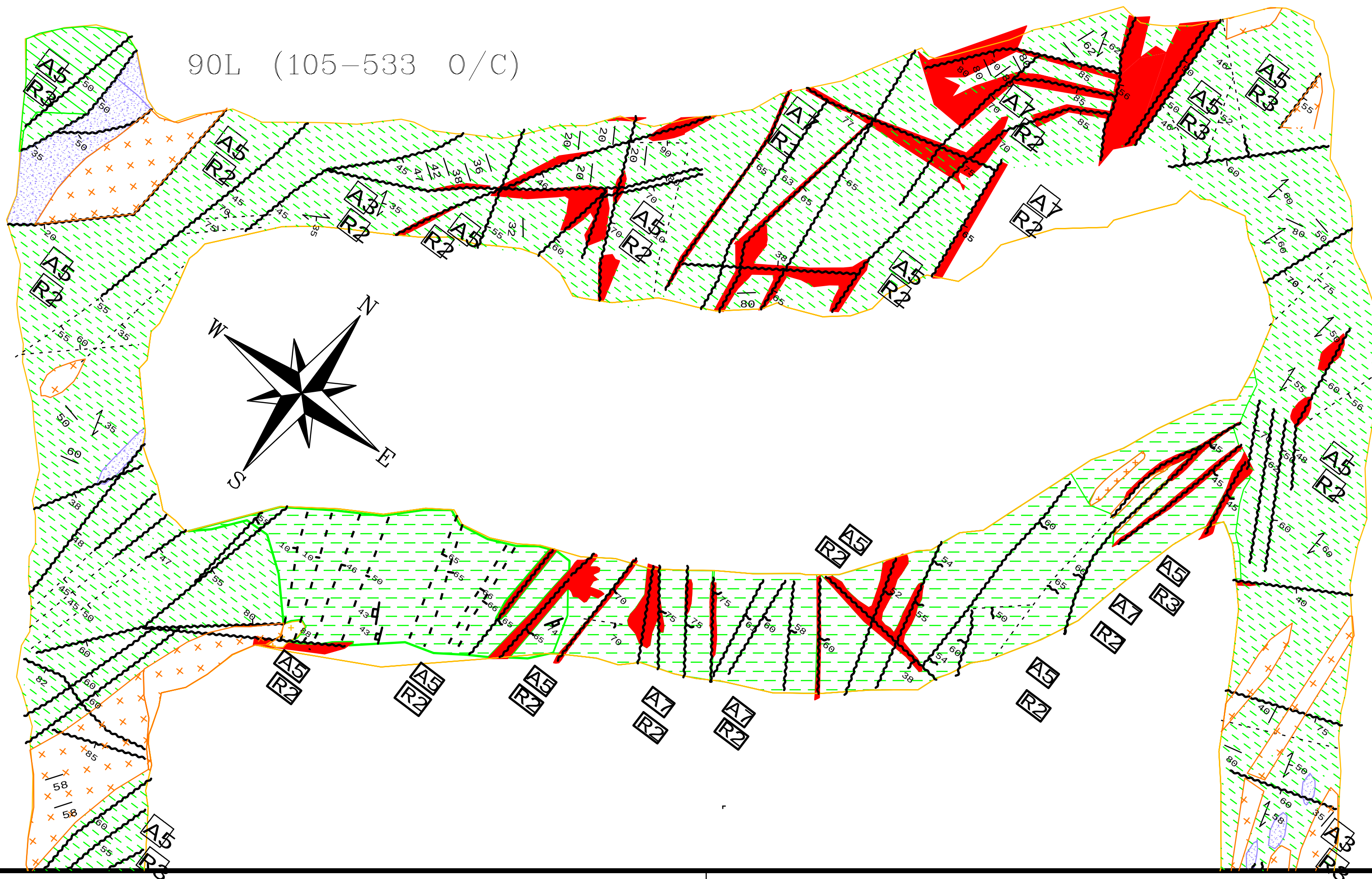
TREND DEV. = 065°

90 LEVEL PLAN

STOP 6: "03" STOPE ON 90mL, 105-513 O/C.

LOOKING NORTHEAST AT FACE. CROWN IN BACK. RMR IS IN EXCESS OF 65% IN WALLS AND 55% IN BACK DUE TO FLAT JOINTS. ORE DRIFTS 4.3m X 4.3m IN DIMENSION. HW HAS PARALLEL JOINTS. DRY SOME AREAS MOIST. 50m CROWN ABOVE BACK. NOTE JUST GETTING INTO ORE WILL BE LOWER (AP) MAY APPROACH A

22



TITLE:

EAGLE POINT MINE

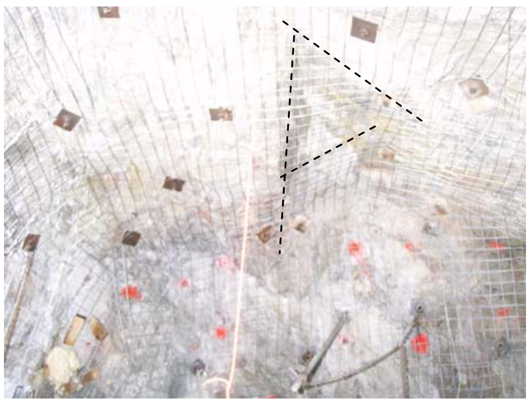
90L GEOLOGICAL MAPPING

DATE:	24-Nov-2010	DWG #:	
SCALE:	1:250	REVISION:	
BY:	KF	FILE:	
REVISED BY:			

Cameco

RABBIT LAKE OPERATION
EAGLE POINT MINE

WALL RMR IS 65% - 55% IN PROXIMITY OF SHEAR/VUGS.



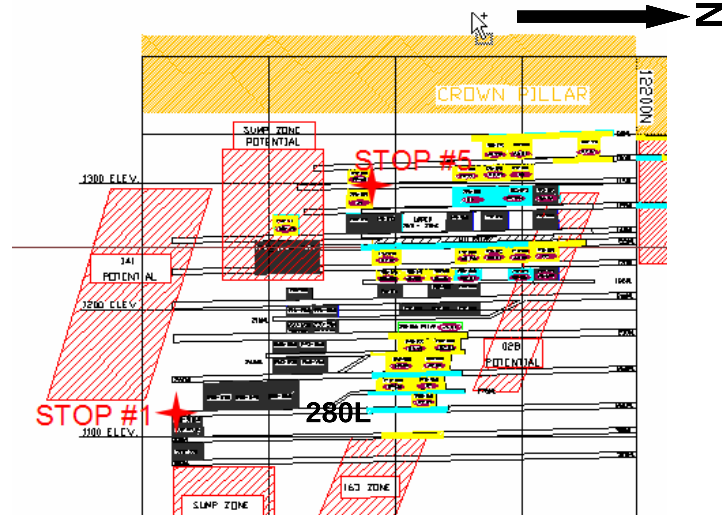
BACK RMR IS 65% - NO FLAT JOINTS OBSERVED

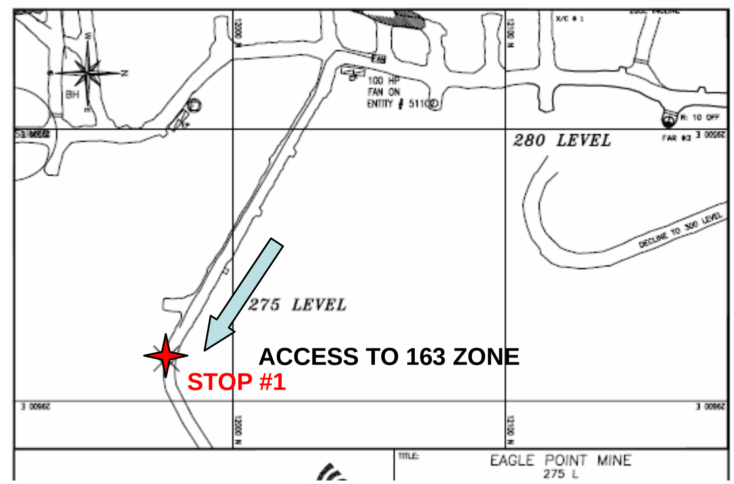
TREND OF TUNNEL IS 120º (MINE NORTH)

RMR BACK/WALL GNEISS

1) STRENGTH	100-200MPa	12
2) RQD	90%-75%	17
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT-SLT	20-6
5) GRNWTR	MOD PRESS	4
	RATING	68%-54%
STRUCTURE	NO FLAT	
	DESIGN (BACK)	64%+

RMR IS IN EXCESS OF 65% HOWEVER IN PROXIMITY OF SHEAR/VUGS HAVE OPEN JOINTS 5mm+. APPROACHING 55% RMR

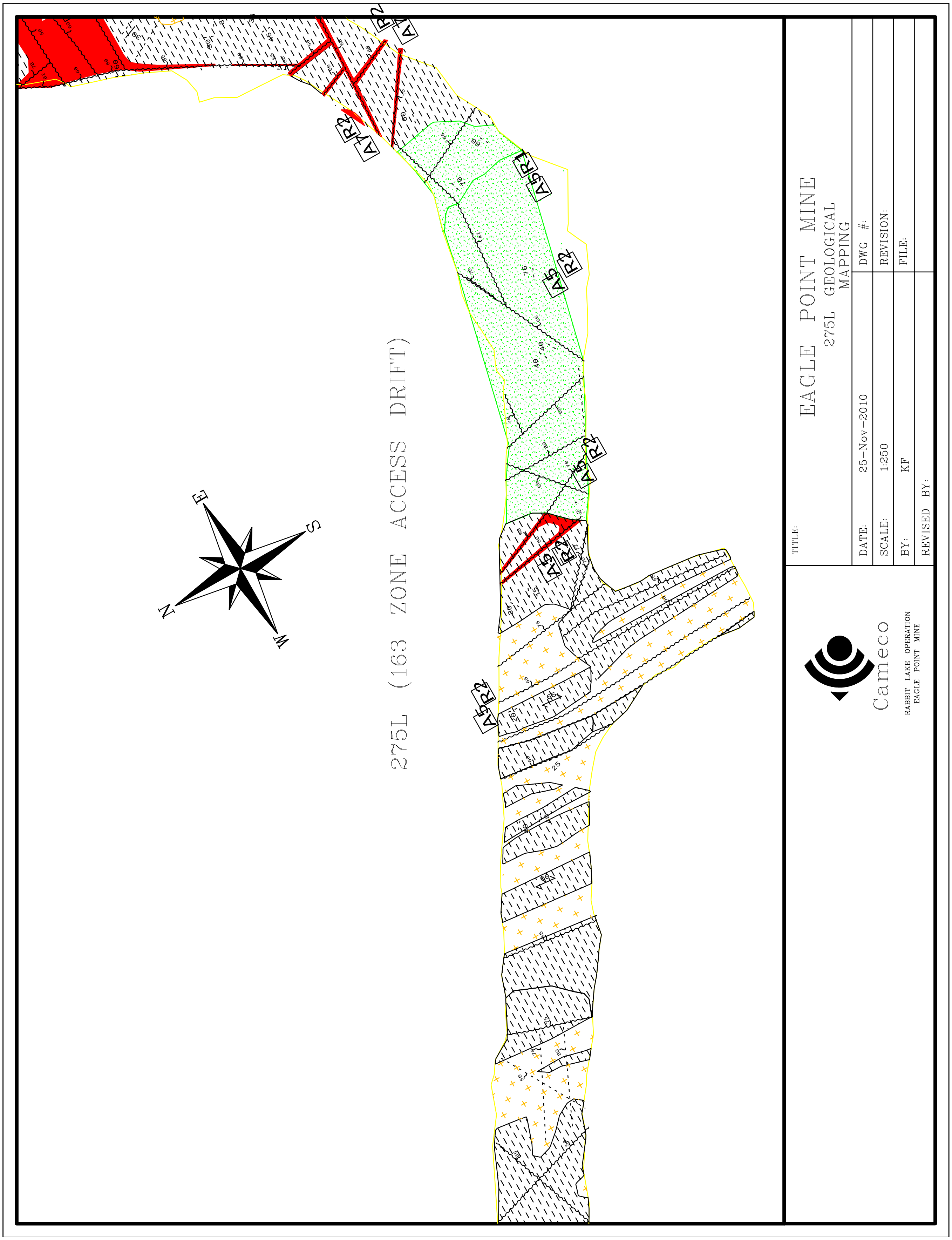




STOP #1: 280L, ACCESS TO 163 ZONE

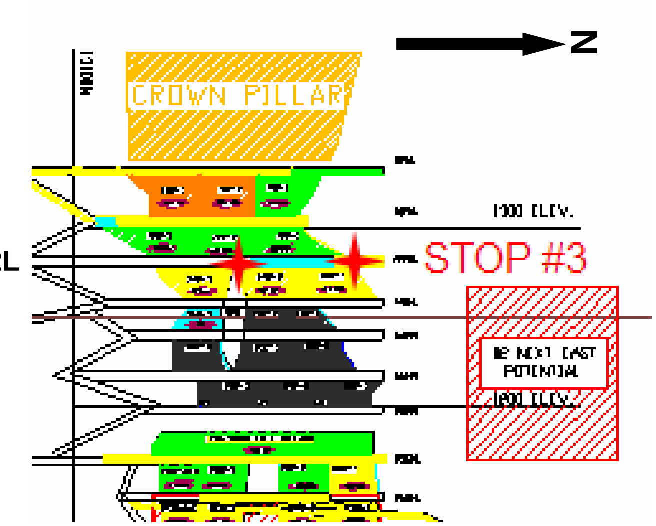
LOOKING EAST AT DEVELOPMENT FACE. THIS AREA IS DRIVEN FOR ACCESS TO 163 ZONE ON 280L WHICH IS ~20m AWAY. CONCERN IS WATER AS HAVE QUARTZITE/FAULT CONDUIT WHICH CARRIES WATER (JR). HEAD HAS BEEN RECORDED TO 140L AND UPON DEPRESSURIZING ON 340L RESULTED IN WATER LEVEL DROPPING TO BELOW 170L. SAME WATER BEARING STRUCTURE RECORDED ON 400L/340L.PRESSURES ON 340L ARE 2MPa (285psi) ~ 200m HEAD. NOMINAL DRIFT SIZE IS 4.3m X 4.3m AND MEASURED IS 5.1m (W) X 4.5m (H). MINERALIZED ZONES OBSERVED (CARBONATE). OBSERVED WEDGES IN BACK (SMALL).

2


275L (163 ZONE ACCESS DRIFT)
N
E
S
W
A5R2
A5R1
A1R2
TITLE:
EAGLE POINT MINE
275L GEOLOGICAL
MAPPING
DATE: 25-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE

RMR BACK/WALL GNEISS		
1) STRENGTH	100-200MPa	12
2) RQD	90%-75%	17
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT	20
5) GRNWTR	MOIST	7
	RATING	74%
STRUCTURE	MINOR FLAT	-10%
	DESIGN (BACK)	64%+

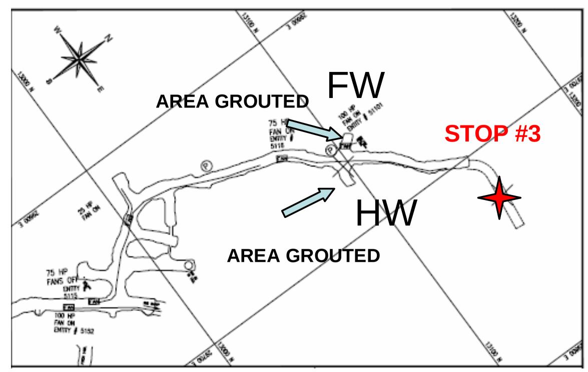
122L



LOOKING AT FACE – DEVELOPMENT WASTE. THIS IS NOT IMMEDIATE HW.

BACK OF DEVELOPMENT

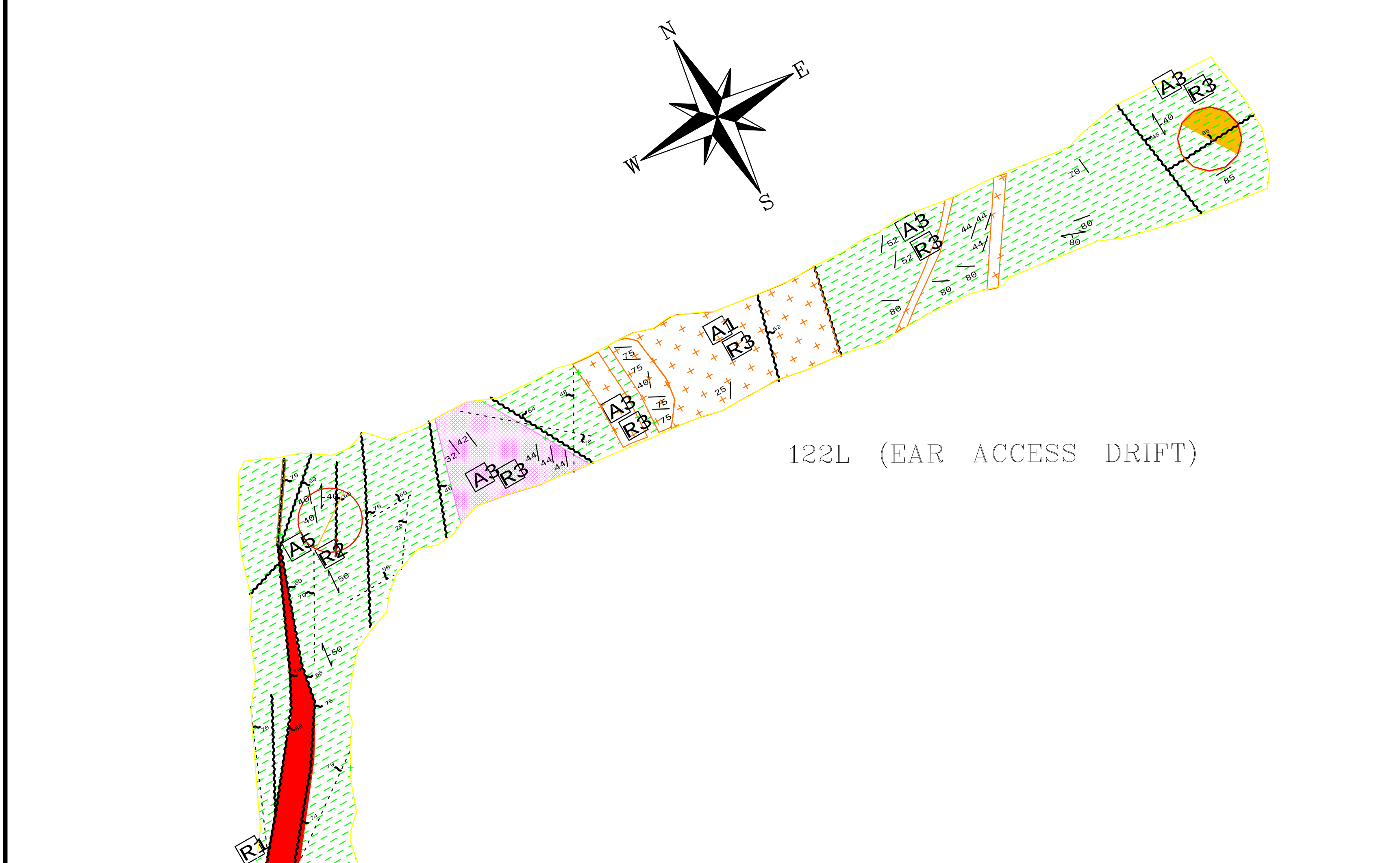
SCREEN IN CORNERS/HAUNCHES SHOULD HAVE OVERLAP (JH/RP)



PLAN 122L

STOP #3: 122L, 02 NEXT ACCESS

LOOKING EAST AT FACE. THIS IS THE UPPER MOST DEVELOPED LEVEL FOR “02 NEXT STOPE” – MINING IS 20m BELOW. DEVELOPMENT IS SHOTCRETED DUE TO RADON. AREA IS GROUTED TO CONTROL WATER INFLOW. THIS IS A WASTE HEADING THAT WENT THROUGH ORE THAT HAS BEEN SHOTCRETED WHICH PREVIOUSLY HAD HIGH WATER FLOW/RADON AND SUBSEQUENTLY GROUTING RESULTED IN MINOR WATER. SUPPORT IS 2.4m(8ft) LONG MECHANICAL BOLTS WITH 1.5m(6ft) SS39 SPLITSETS ON WALL ON 4ft X 4ft WALL + 9GAUGE WELD MESH SCREEN. NOTE SCREEN IN HAUNCHES TO WITHIN 5ft OF FLOOR. ISOLATED AREAS NO SCREEN. NOTE SCREEN GENERALLY OBSERVED TO WITHIN 5ft OF FLOOR. 4



Cameco

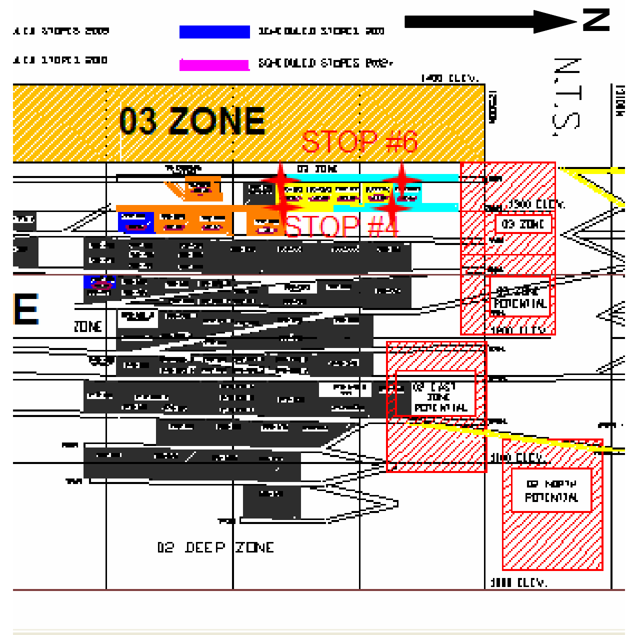
RABBIT LAKE OPERATION
EAGLE POINT MINE

TITLE:	EAGLE POINT MINE 122L GEOLOGICAL MAPPING
DATE: 25-Nov-2010	DWG #:
SCALE: 1:250	REVISION:
BY: KF	FILE:
REVISED BY:	

RMR WALL GNEISS		
1) STRENGTH	100-200MPa	12
2) RQD	90%-75%	17
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT	20
5) GRNWTR	DRY-SEVERE	10-0
STRUCTURE	RATING	74%-64%
	DESIGN	65%

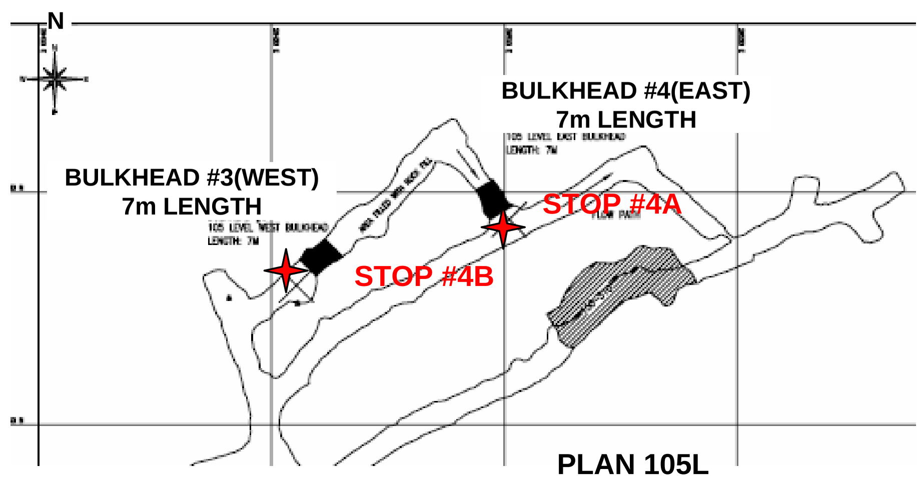
NOTE Q'=(RQD/Jn)* (Jr/Ja)
=(80/6)*(1.5/1.5)=13.3

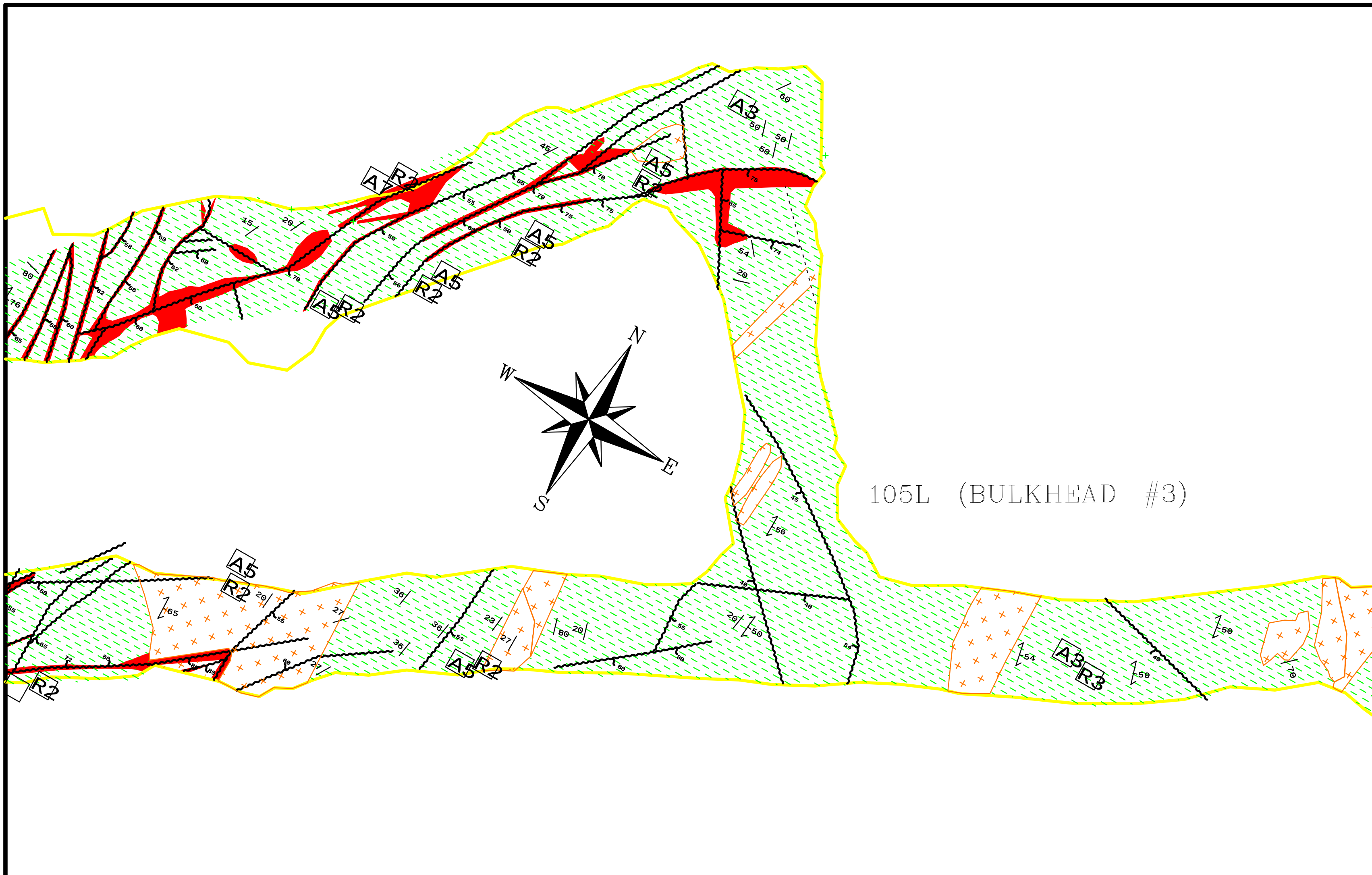
122L



BULKHEAD #4: FLOW RATE IS 120m^3/hr (528gpm – US). WATER ADDED FROM ALL OTHER SOURCES (ENTIRE MINE) = 242m^3/hr SO TOTAL EQUALS 361m^3/hr (1590gpm – US). **NOTE IS EQUIVALENT TO THREE NQ DRILLHOLES (120m^3/hr PER DRILL HOLE X 3) OBSERVATION BY J.H.**

STOP #4A: 105L, 03 STOPE (BARRICADE #3 EAST)





TITLE:	EAGLE POINT MINE 105L GEOLOGICAL MAPPING		
DATE:	25-Nov-2010	DWG #:	
SCALE:	1:250	REVISION:	
BY:	KF	FILE:	
REVISED BY:			

Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE

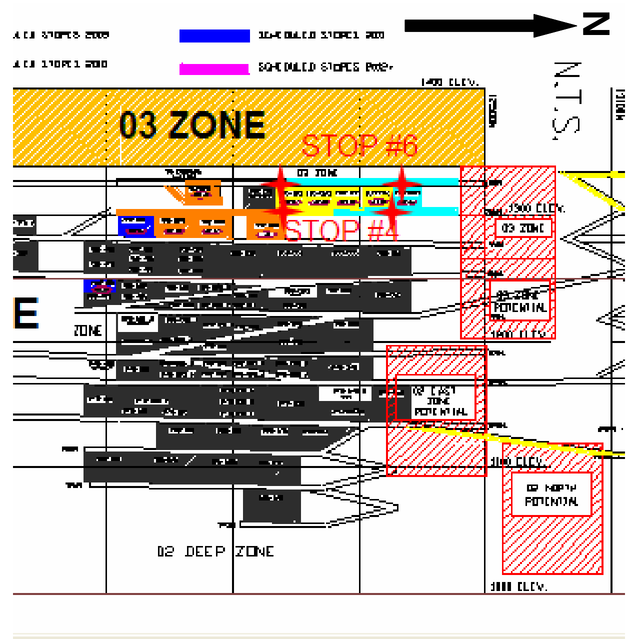
BULKHEAD #3 (WEST): DRY – NO FLOW OBSERVED AS THIS BULKHEAD IS FEW METERS ABOVE #4.

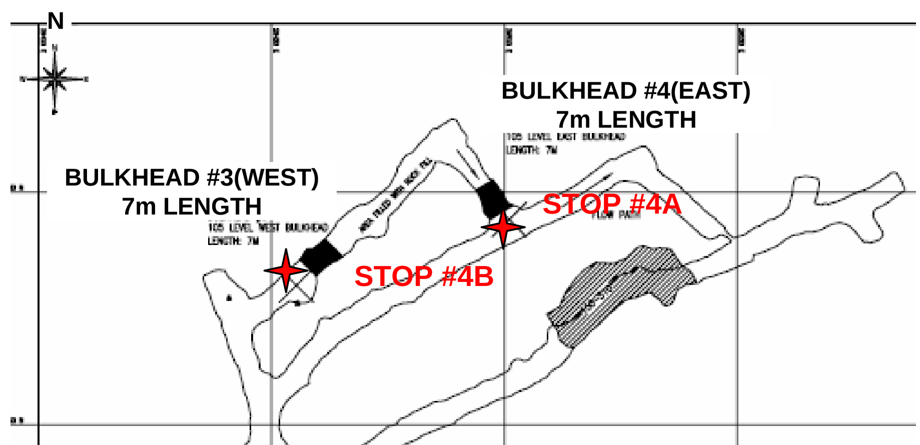
RMR WALL GNEISS		
1) STRENGTH	100-200MPa	12
2) RQD	90%-75%	17
3) SPACING	50-300mm+	15
4) CONDITON	SLT	12
5) GRNWTR	DRY-SEVERE	4-0
STRUCTURE	RATING	60%-56%
	DESIGN	55%

JS	DIP	DDR
1	70°	090°
2	58°	218°
3	72°	320°
4	42°	149°

* RELATIVE TO MINE NTH

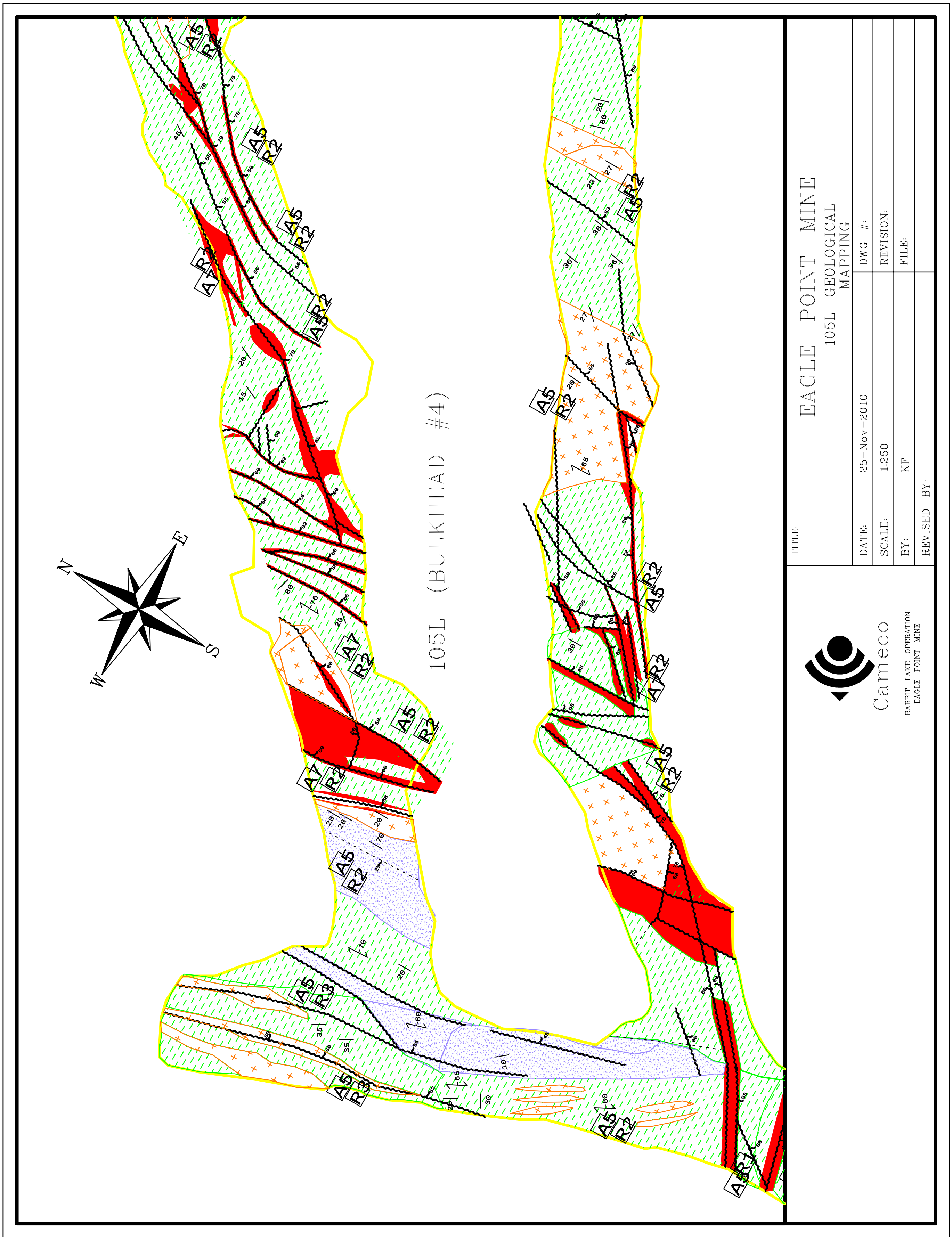
122L





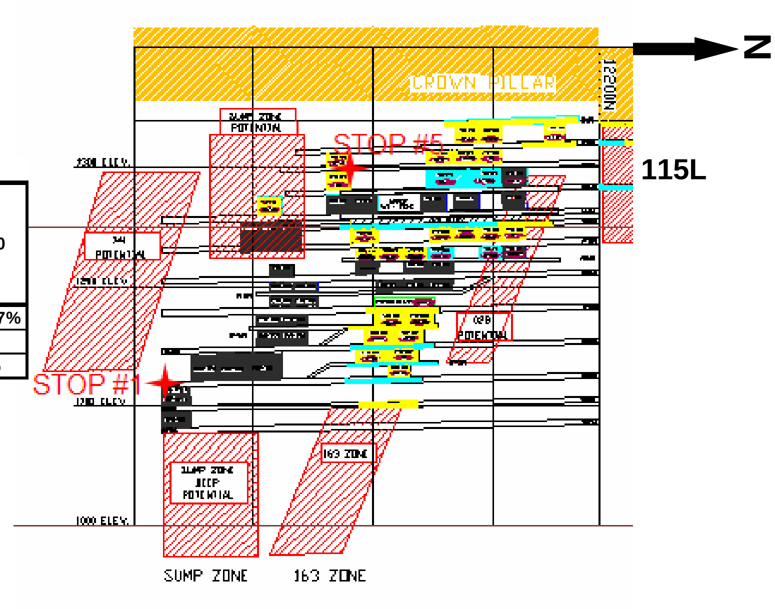
PLAN 105L

STOP #4B: 105L, 03 STOPE (BARRICADE #3 WEST)


105L (BULKHEAD #4)
TITLE:
EAGLE POINT MINE
105L GEOLOGICAL MAPPING
DATE: 25-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE

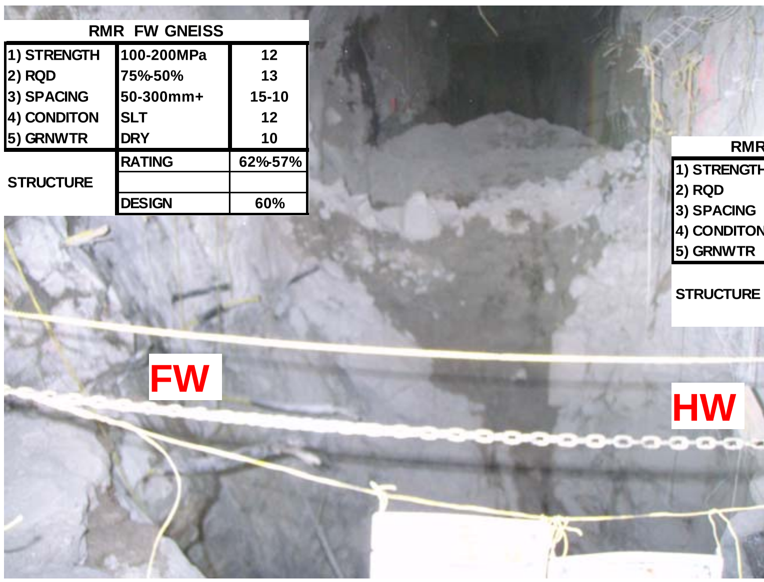
RMR FW GNEISS

1) STRENGTH	100-200MPa	12
2) RQD	75%-50%	13
3) SPACING	50-300mm+	15-10
4) CONDITON	SLT	12
5) GRNWTR	DRY	10
STRUCTURE	RATING	62%-57%
	DESIGN	60%



RMR FW GNEISS

1) STRENGTH	100-200MPa	12
2) RQD	75%-50%	13
3) SPACING	50-300mm+	15-10
4) CONDITON	SLT	12
5) GRNWTR	DRY	10
STRUCTURE	RATING	62%-57%
	DESIGN	60%



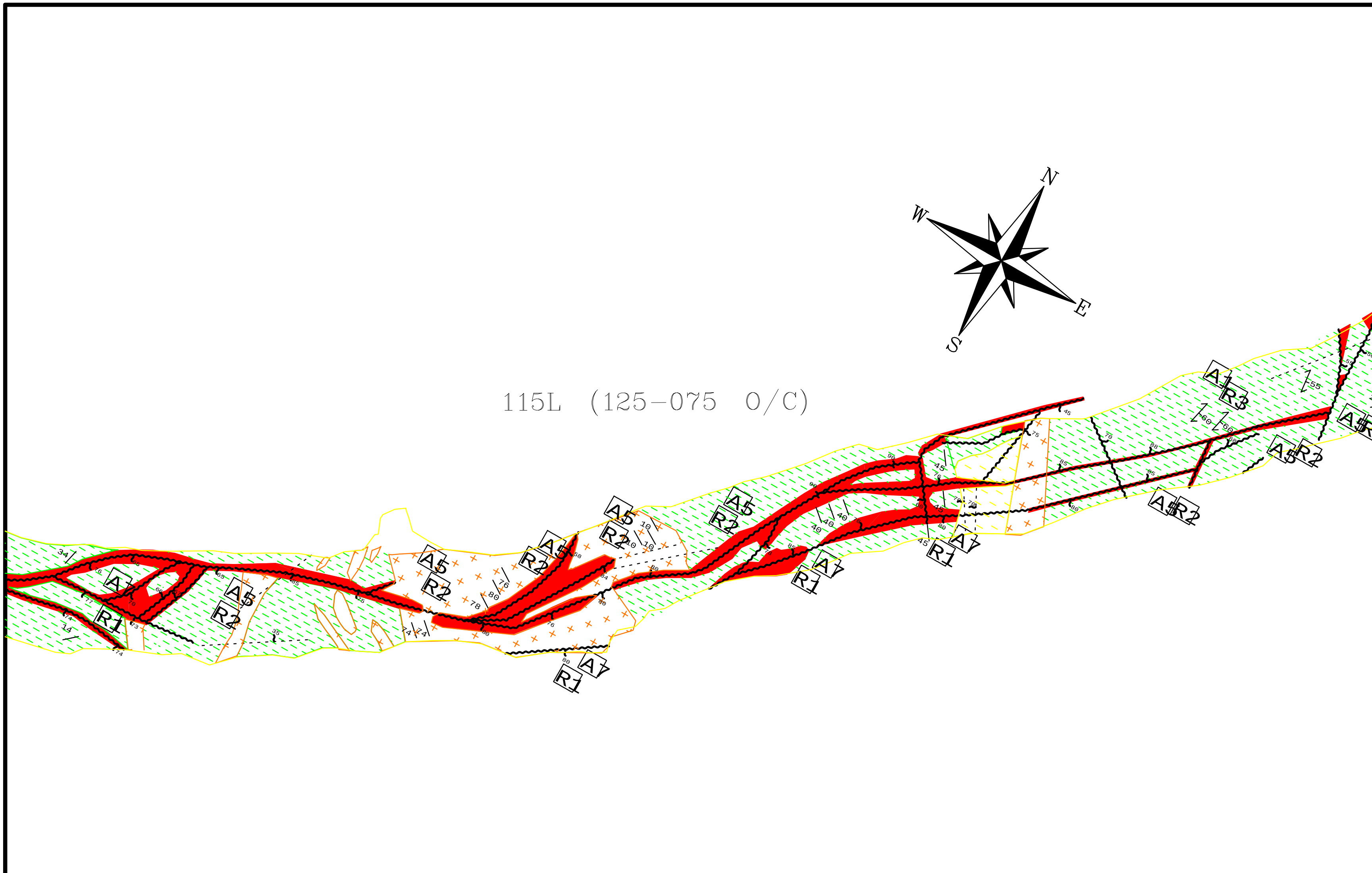
RMR HW/BACK GNEISS

1) STRENGTH	100-200MPa	12
2) RQD	90%-75%	17
3) SPACING	50-300mm+	15
4) CONDITON	SLT	12
5) GRNWTR	DRY	10
STRUCTURE	RATING	66%
	FLAT	-10%
	DESIGN(BACK)	56%

STOP #5: 115L, 125-075 STOPE O/C (163 ZONE)

LOOKING NORTHEAST. LONGHOLE STOPE FROM 115-125L. LOOKING AT SLOT WHICH IS ~9m LENGTH X 18m HT. HAD FW SHEAR WITHIN PREVIOUS STOPE WHICH RESULTING IN UNDERCUT (~5m+) OF FW (REFER CMS) WHICH RESULTS IN CONCERN MINING ABOVE TO THE LEVEL TO 100L. THE FW HAS BEEN CABLED WITHIN SLOT AREA TO NEGATE EFFECTS. BACK SPAN IS 5.4m HAVING AN RMR OF 55% DESIGN. SUPPORT IS IN FORM OF 2.4m(8ft) REBAR ON 1.2m X 1.2m PATTERN + CABLES IN BACK/WALLS TO CONFINE WALL SLOUGH. FW IS LOCALLY CABLED DUE TO FW SHEAR OBSERVED FROM PREVIOUS STOPE.

7

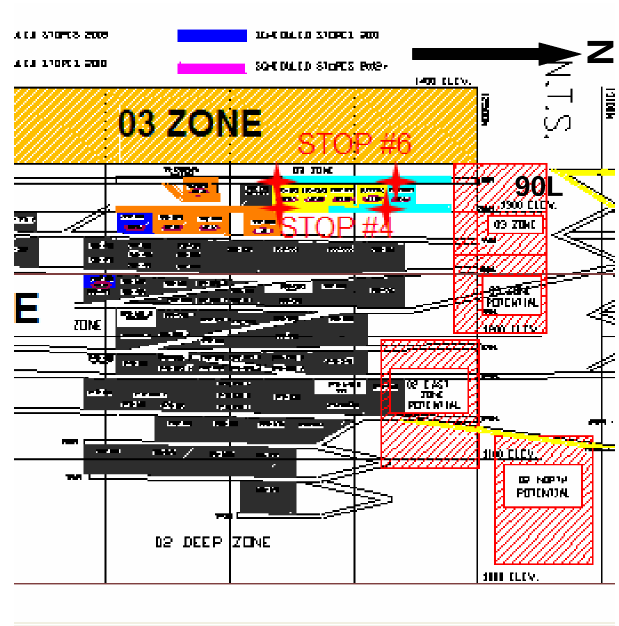


TITLE: EAGLE POINT MINE 115L GEOLOGICAL MAPPING			
DATE:	25-Nov-2010	DWG #:	
SCALE:	1:250	REVISION:	
BY:	KF	FILE:	
REVISED BY:			

Cameco

RABBIT LAKE OPERATION
EAGLE POINT MINE

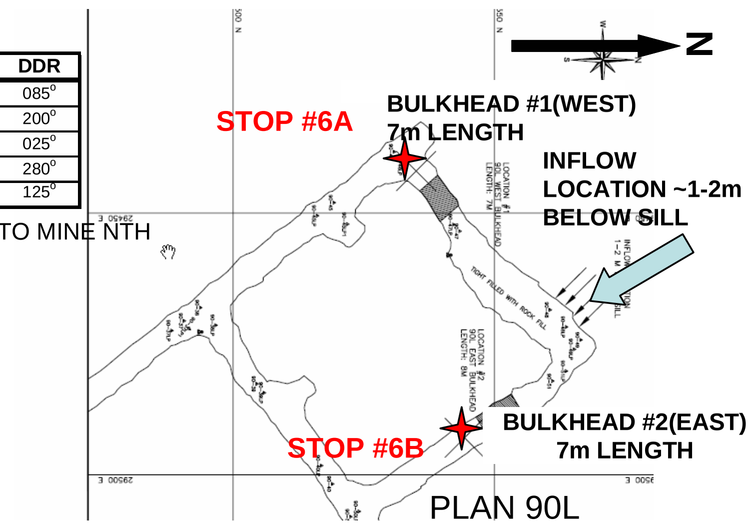
RMR	WALL GNEISS	
1) STRENGTH	100-200MPa	12
2) RQD	90%-75%	17
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT-SLT	20-12
5) GRNWTR	MOD-SEVERE	4-0
	RATING	68%-56%
STRUCTURE		
	DESIGN	60%



BULKHEAD #1: PROXIMITY OF WALL HAVING 60% RMR LARGELY DUE TO SLIGHT JOINT OPENNESS. NO FLOW.

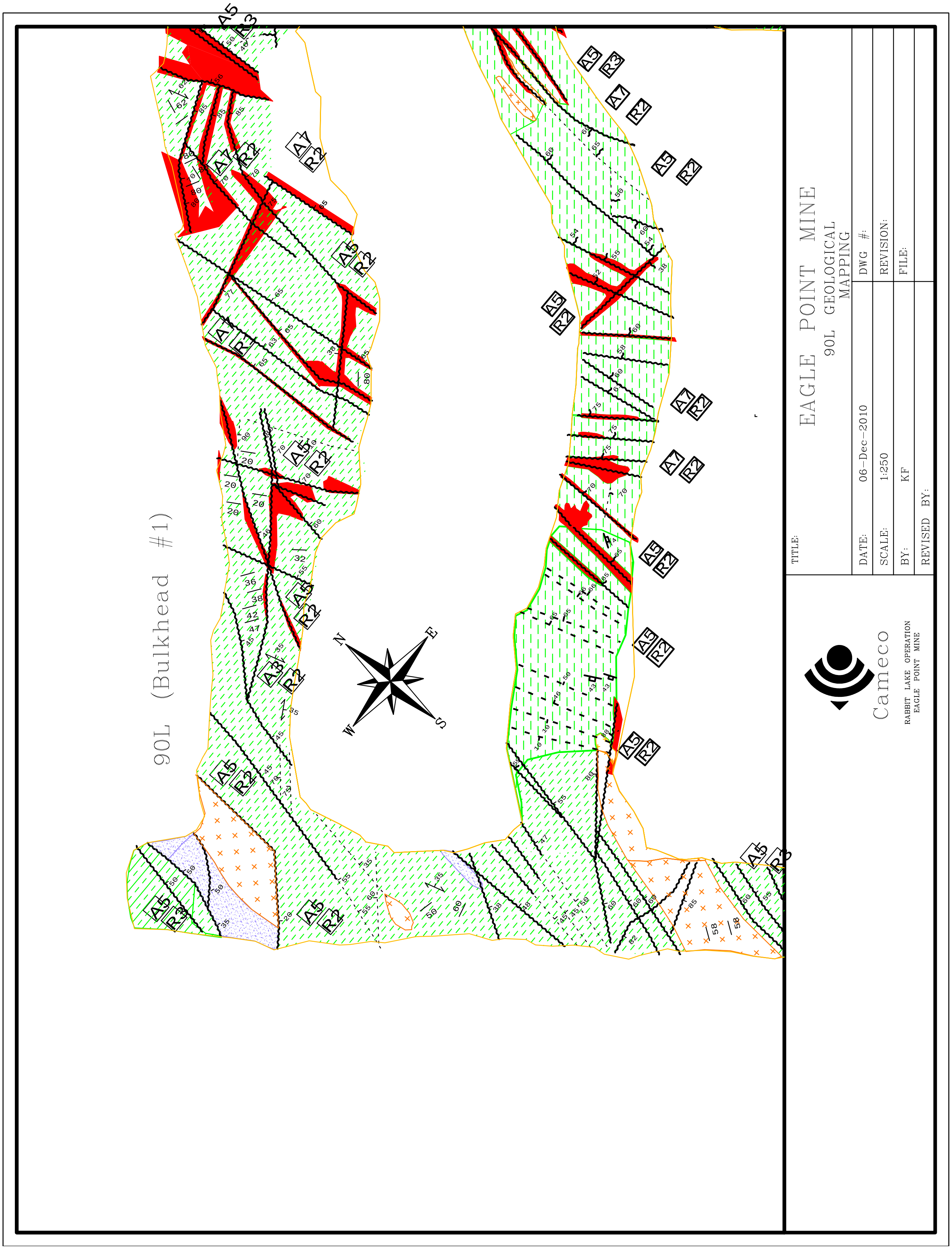
JS	DIP	DDR
1	70°	085°
2	60°	200°
3	80°	025°
4	30°	280°
5	70°	125°

* RELATIVE TO MINE NTH

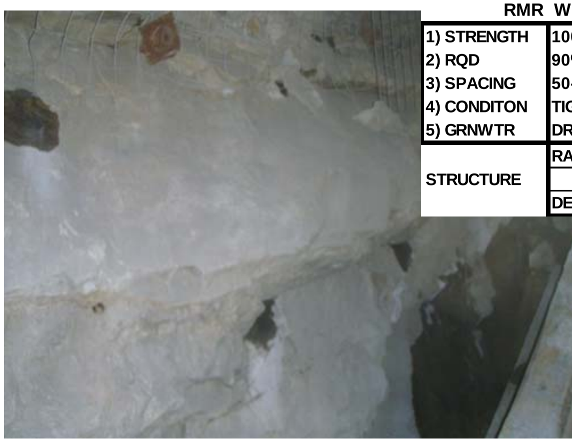


STOP #6A: 90L, 03 STOPE (BARRICADE #1 WEST)

8



90L (Bulkhead #1)
TITLE:
EAGLE POINT MINE
90L GEOLOGICAL MAPPING
DATE: 06-Dec-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE

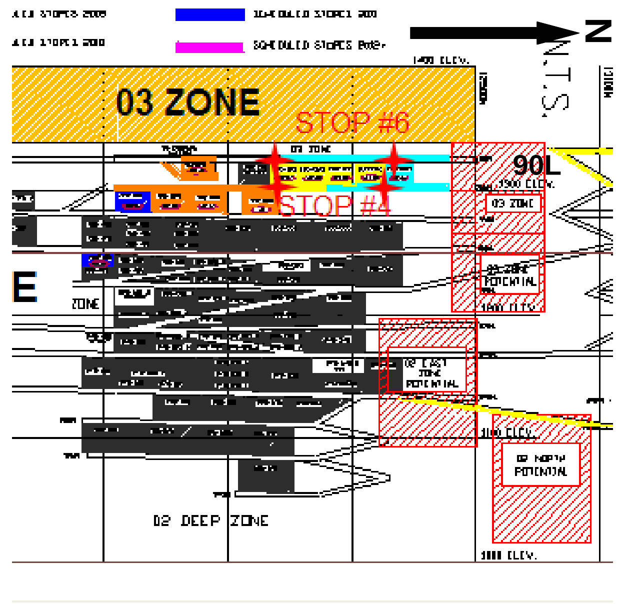


RMR WALL GNEISS		
1) STRENGTH	100-200MPa	12
2) RQD	90%-75%	17
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT	20
5) GRNWTR	DRY	10
STRUCTURE	RATING	74%
	DESIGN	74%

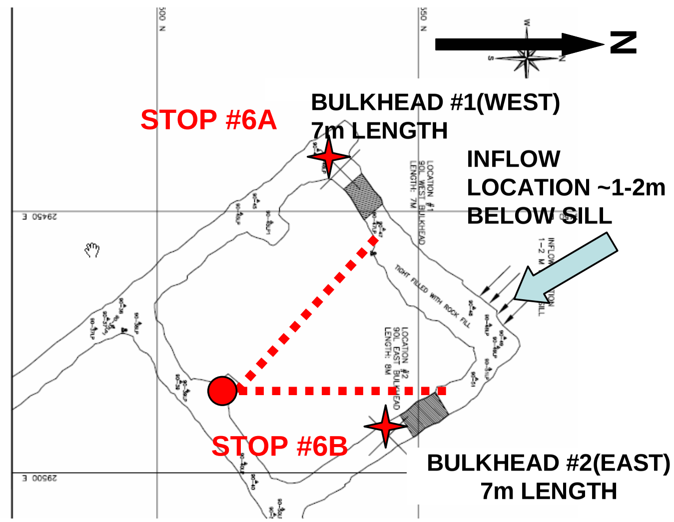
JS	DIP	DDR
1	65°	083°
2	35°	225°
3	85°	045°
4	60°	065°
5	25°	185°

* RELATIVE TO MINE NTH

BULKHEAD #2: PROXIMITY OF WALL HAVING 74% RMR (TIGHT JOINTS/DRY). NO FLOW



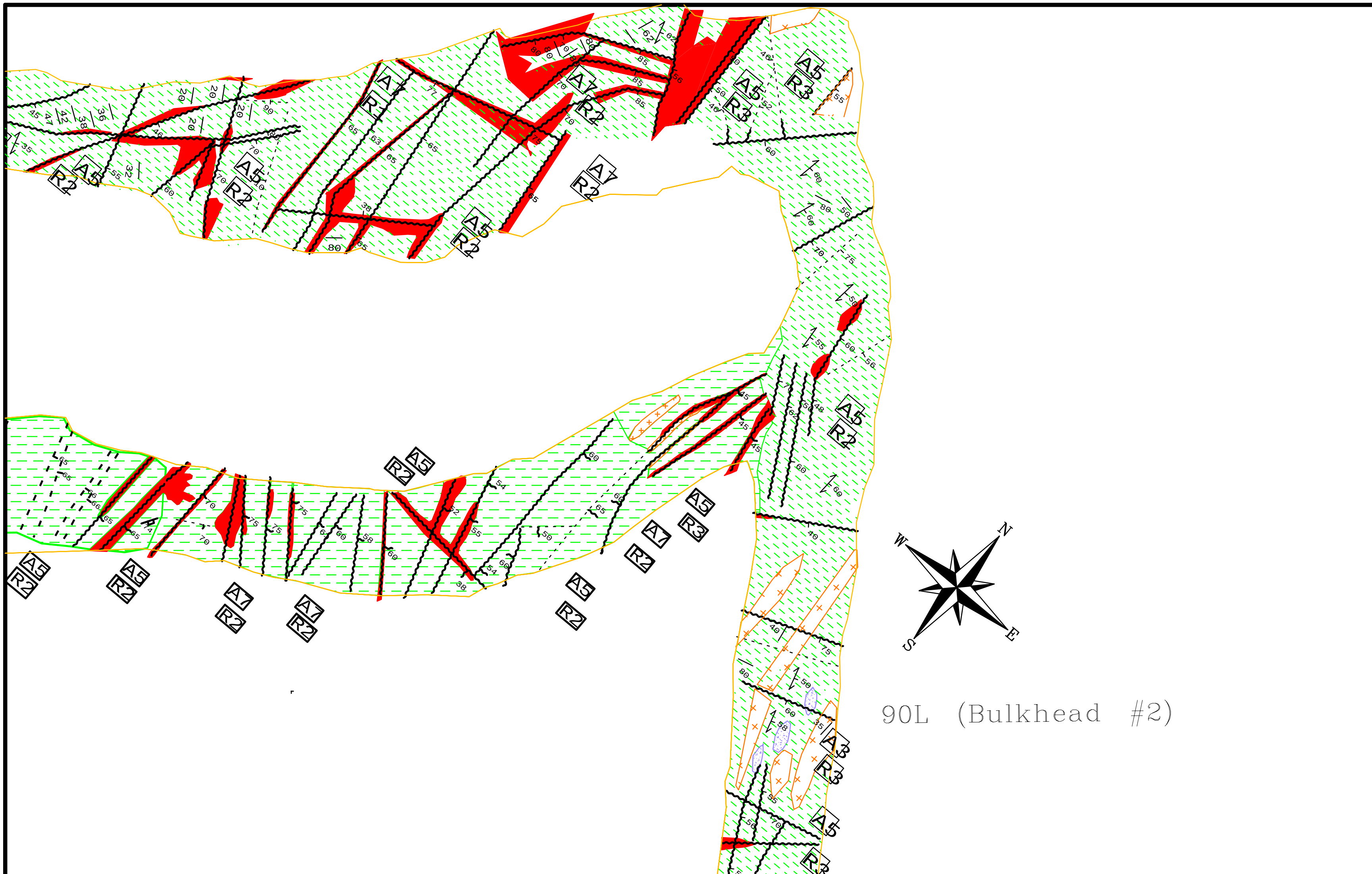
ADJACENT DRILLING OF FOUR(4) RELIEF HOLES FOR BULKHEADS.



PLAN 90L

9

STOP #6B: 90L, 03 STOPE (BARRICADE #2 EAST)



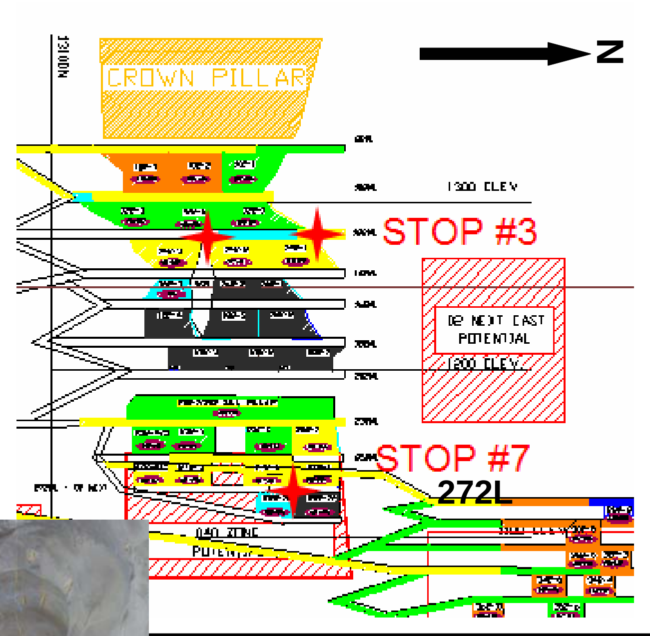
TITLE: EAGLE POINT MINE
90L GEOLOGICAL MAPPING

DATE:	06-Dec-2010	DWG #:
SCALE:	1:250	REVISION:
BY:	KF	FILE:
REVISED BY:		

Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE

STOP #7A: LOOKING NORTH AT HW CAVE ON 272L WITHIN 292-1A STOPE.

PILLAR BETWEEN 7B AND 7C ~3-5m WIDE.

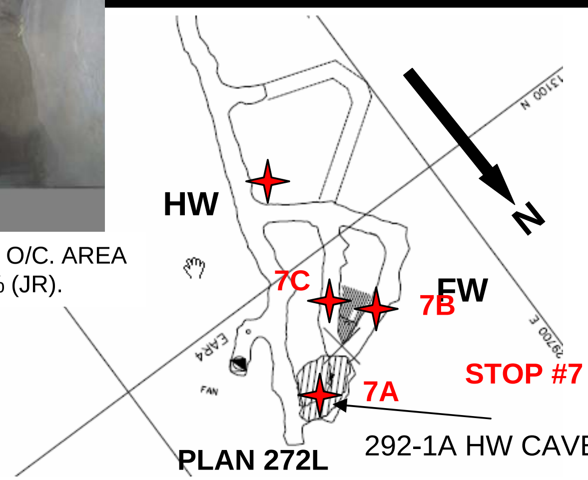


RMR FW GNEISS

1) STRENGTH	100-200MPa	12
2) RQD	90%-75%	17
3) SPACING	50-300mm+	15-10
4) CONDITON	TIGHT-SLT	20-12
5) GRNWTR	DRY	10
	RATING	74%-61%
STRUCTURE	FLAT	-10%
	DESIGN	55%

STOP #7B: FW SIDE OF 292-1B O/C. AREA IS CABLED. RMR DESIGN ~55% (JR).DUE TO FLAT JOINTS.

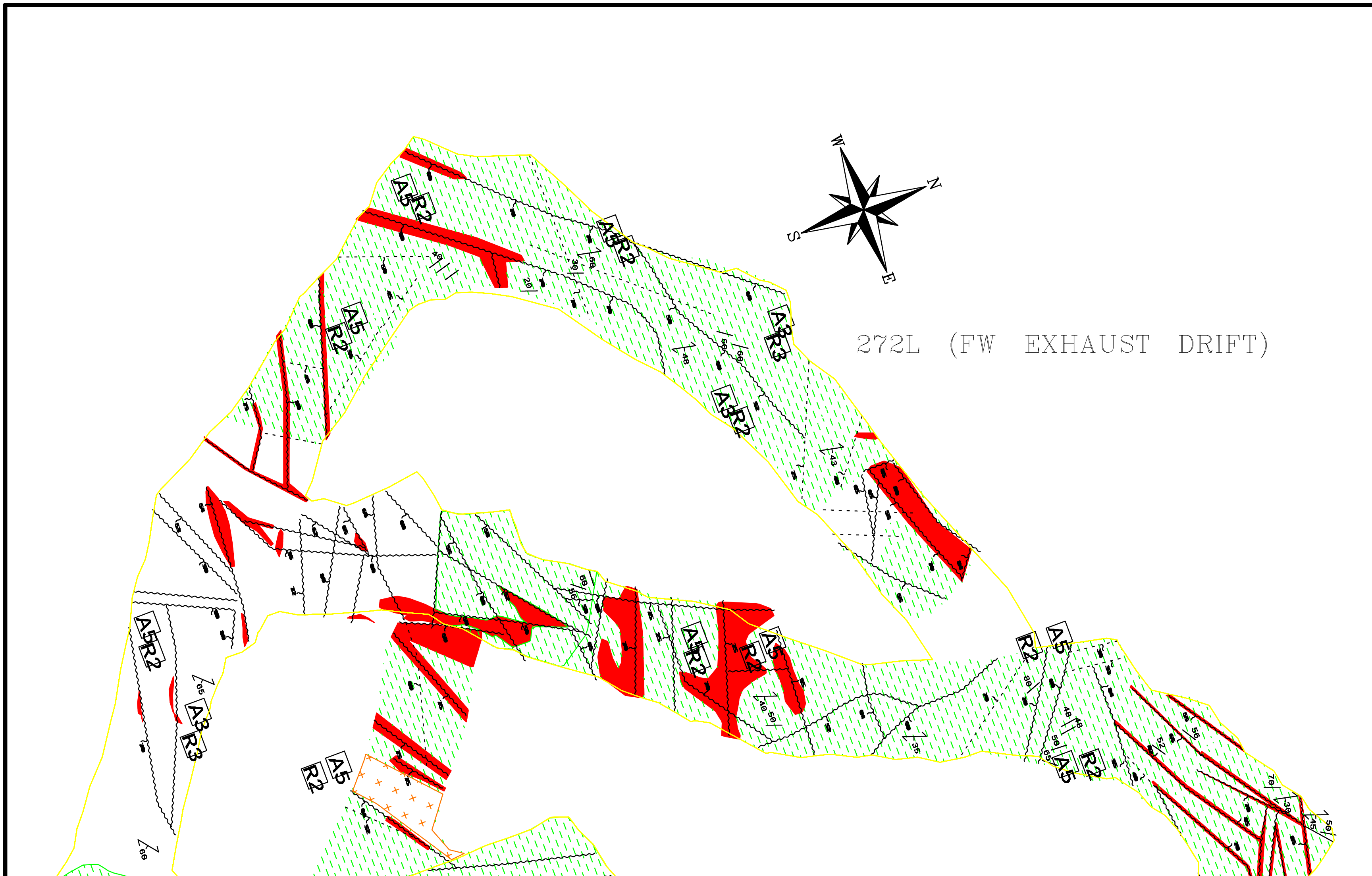
STOP #7C: HW SIDE OF 292-1B O/C. AREA IS CABLED. RMR DESIGN ~55% (JR).



STOP #7: 272L, 292-1A/1B STOPE O/C (LOWER O2 NEXT ZONE)

LOOKING AT CAVE OF 292-1A HW AS WELL AS THE STOPE TO BE MINED NEXT WHICH IS THE 292-1B. THIS STOPE FW AND HW HAVE BEEN CABLED AND INTENT IS TO MINE THE NORTH PILLAR TO A SPAN APPROACHING 15m IN BACK. NOTE THIS IS WITHIN CAVE ZONE FOR SPAN CURVE FOR UNSUPPORTED GROUND. REQUIRE CABLES TO CONFINE 0.5X SPAN POTENTIAL WEDGE IN BACK. NOTE SHOULD BE ~7.5m+1m LENGTH TO PASS BY LARGEST WEDGE APEX. (0.5X SPAN).

10

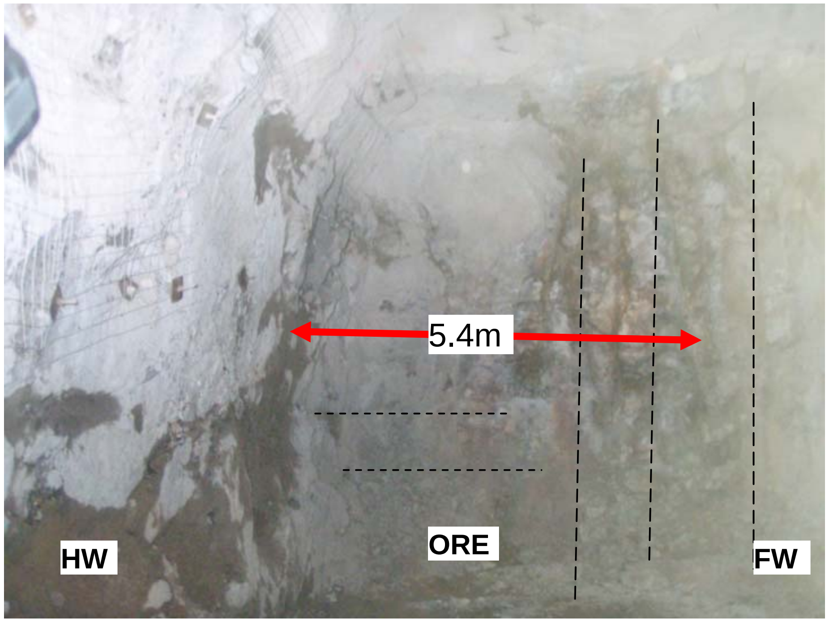


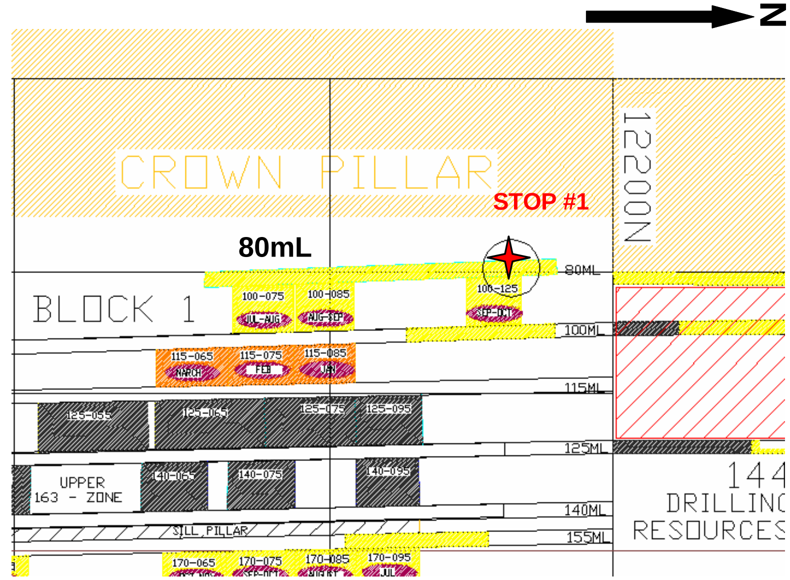
TITLE: EAGLE POINT MINE
272L GEOLOGICAL MAPPING

DATE: 07-Dec-2010
SCALE: 1:250
BY: KF
REVISED BY:

DWG #:
REVISION:
FILE:

Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE





RMR WALL GNEISS

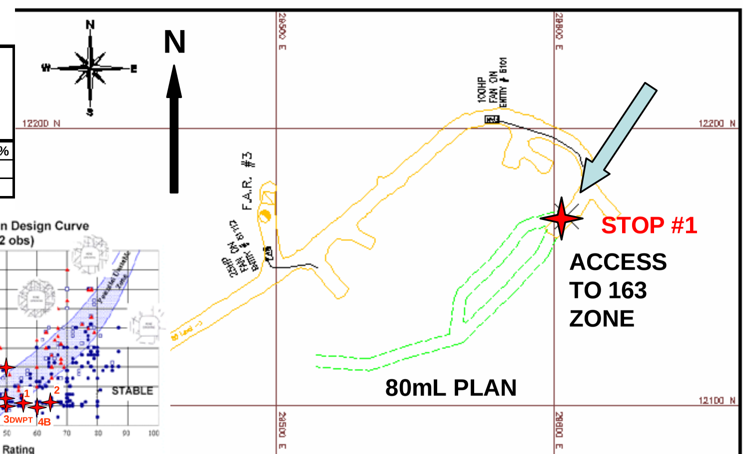
1) STRENGTH	100-200MPa	12
2) RQD	75%-50%	13
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT-SLT	20-12
5) GRNWTR	DRY	10
	RATING	70%-62%
STRUCTURE		
	DESIGN (WALL)	65%+

RMR BACK GNEISS

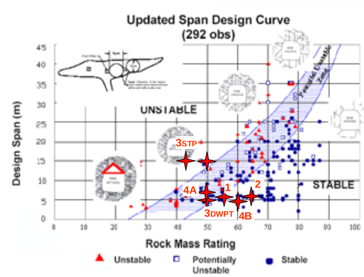
1) STRENGTH	100-200MPa	12-10
2) RQD	75%-50%	13
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT-SLT	20-12
5) GRNWTR	DRY	10
	RATING	70%-60%
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	55%+

RMR WALL GNEISS

1) STRENGTH	100-200MPa	12
2) RQD	75%-50%	13
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT-SLT	20-12
5) GRNWTR	DRY	10
	RATING	70%-62%
STRUCTURE		
	DESIGN (WALL)	65%+

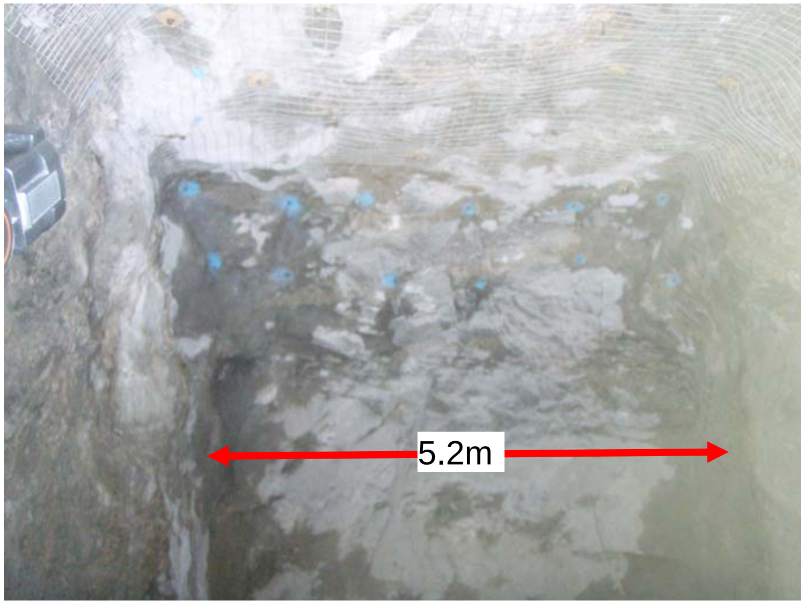


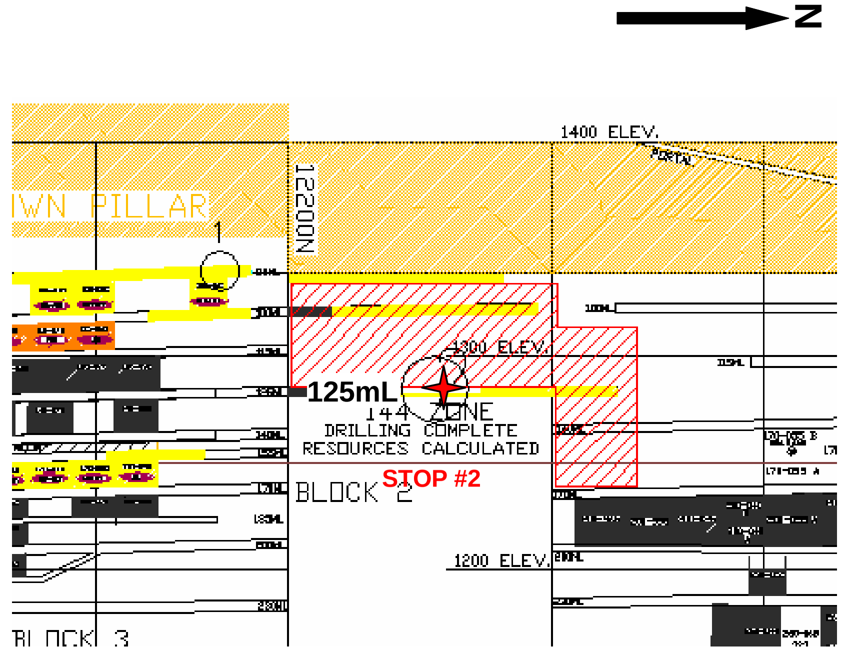
BACK RMR IS 65% - 10%=55%
DUE TO FLAT JOINTS
OBSERVED. ORE IS LOW
ALTERATION WITH VERTICAL
JOINTING. FLAT BACK.



STOP #1: 80L, 163 ZONE (80-100 OVERCUT)

LOOKING SOUTHWEST AT ORE FACE WITHIN 163 ZONE. RMR IS 65% FOR THE FW/HW AND 55% (DUE TO FLAT JOINTS) IN BACK. LOW ALTERATION. SPAN IS 5.4m STABLE WITH 2.4m LONG #7 REBAR ON 1.2m X 1.2m PATTERN WITH 9 GAUGE WWM TO WITHIN 1.5m FROM FLOOR. SHOTCRETE(2") EMPLOYED FOR GAMMA CONTROL. NOTE SCREEN OVER THE SHOTCRETE WITH 9 GAUGE NOT PRIOR TO PLACEMENT OF SHOTCRETE. THE BOTTOM ROW IS SPLITSET(33mm) WITH RESIN REBAR FOR REMAINDER ON 1.2m x 1.2m PATTERN. HW COMPRISED OF PARALLEL JOINTS. ORE HAS VERTICAL JOINTING ALONG WITH MINOR FLAT JOINTING. THE DRIFT HEIGHT IS 5m (MEASURED).

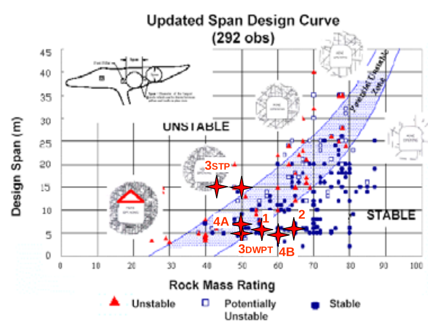


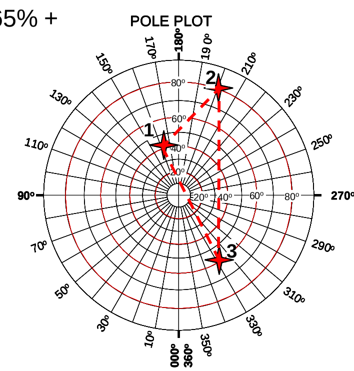


RMR WALL/BACK GNEISS

1) STRENGTH	100-200MPa	12
2) RQD	75%-50%	13
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT-SLT	20-12
5) GRNWTR	DRY	10
	RATING	70%-62%
STRUCTURE	FLAT - MINOR	
	DESIGN (BACK)	65%+

BACK RMR IS 65% + (MINOR FLAT). WALLS RMR IS 65%+. BLOCKY GROUND.

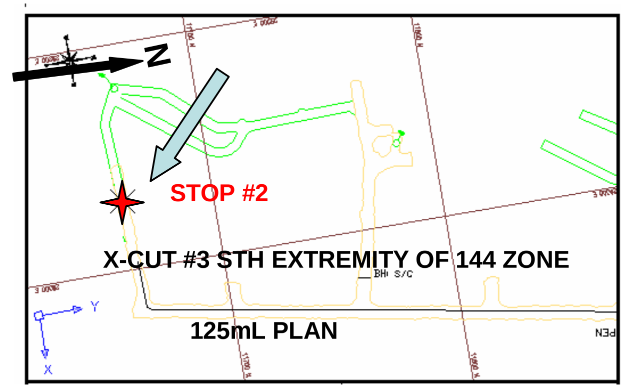




JS	DIP	DDR
1	42°	165°
2	80°	200°
3	60°	330°

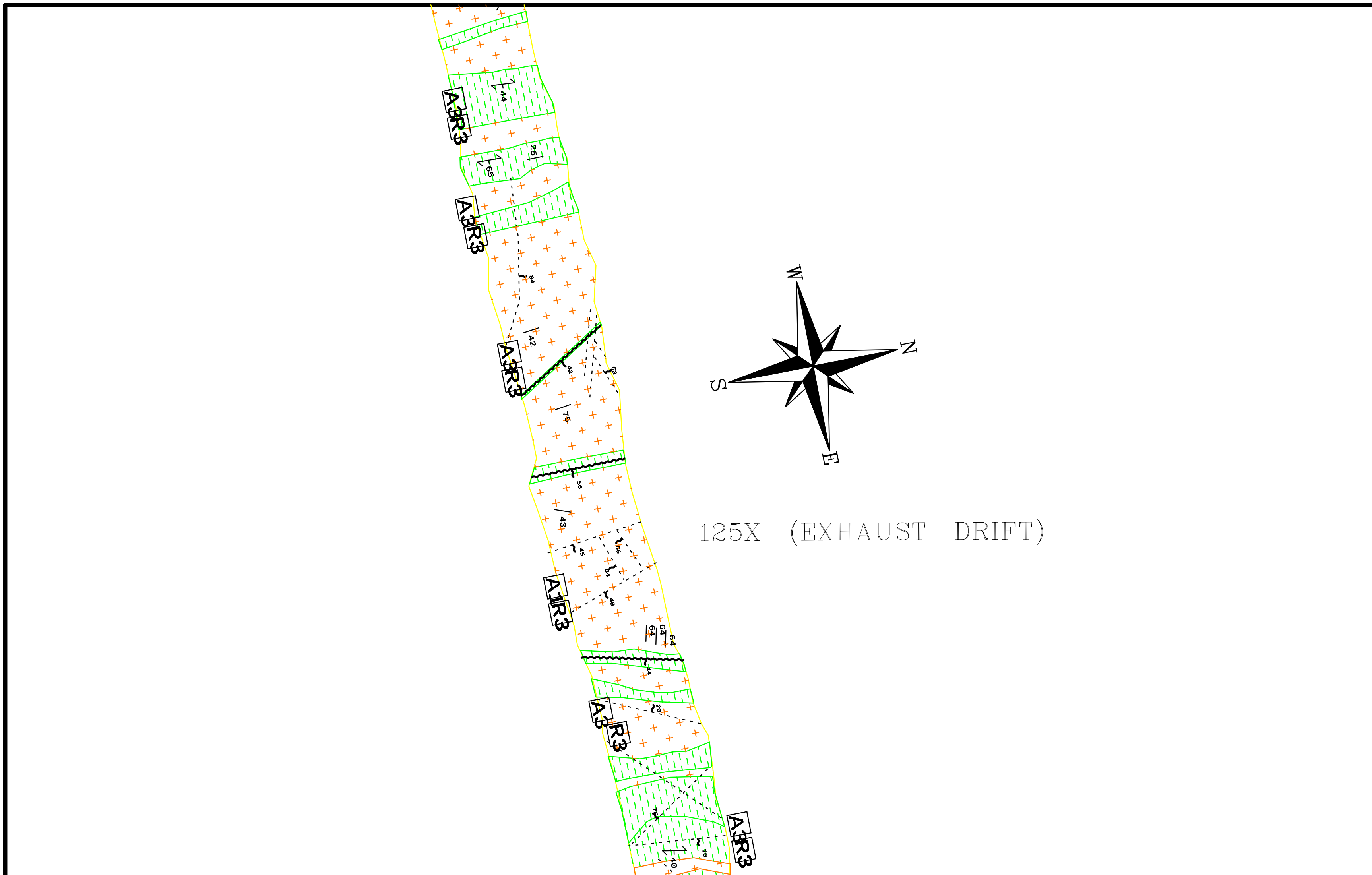
TREND DRIFT= 270°

* NOT "DEAD WEIGHT" WEDGE- SLIDING



STOP #2: 125mL, X-CUT #3 STH EXTREMITY OF 144 ZONE (WASTE DEVELOPMENT)

LOOKING WEST AT DEVELOPMENT FACE COMPRISED OF GNEISS AND PEGMATOID. WASTE DEVELOPMENT TOWARDS 144 ZONE. SUPPORT COMPRISED OF 2.4m MECHANICAL BOLTS(5/8") ON 1.2m X 1.2m PATTERN WITH 1.8m SPLITSETS (33mm) ON WALLS ON 1.2m X 1.2m.



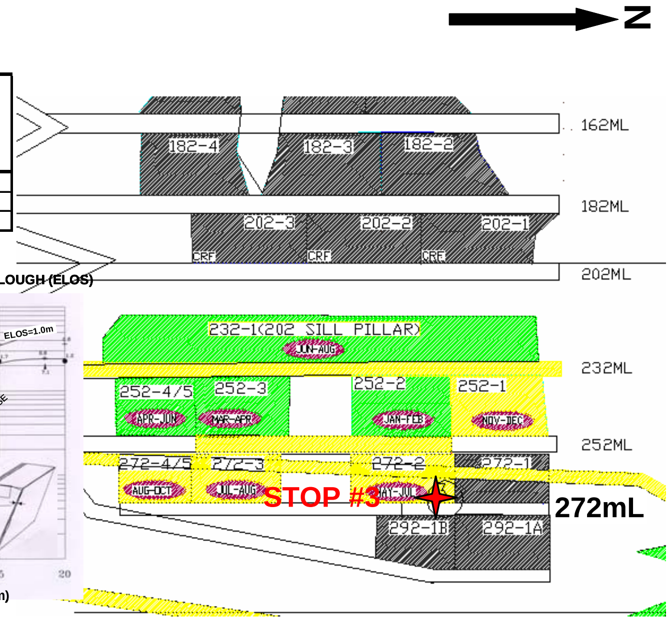
Cameco

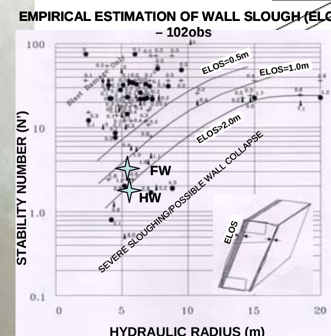
RABBIT LAKE OPERATION
EAGLE POINT MINE

TITLE: EAGLE POINT MINE 125X GEOLOGICAL MAPPING	
DATE: 25-Nov-2010	DWG #:
SCALE: 1:250	REVISION:
BY: KF	FILE:
REVISED BY:	

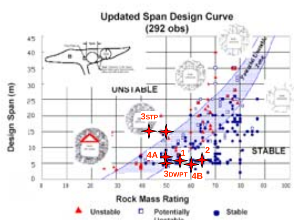
RMR BACK GNEISS

1) STRENGTH	100-200MPa	12
2) RQD	75%-50%	13
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT-SLT	12
5) GRNWTR	DRY	10
	RATING	62%
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	52%

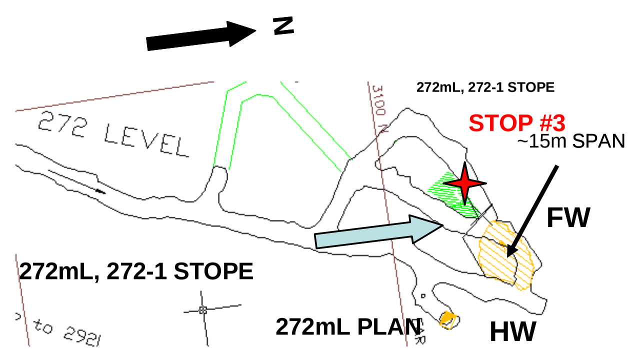




LOOKING NORTH AT DWPT TOWARDS 272-1 STOPE. BACK RMR IS 50% HOWEVER IN STOPE MORE ALTERED (JR) AND MOST LIKELY 45% RMR. NOTE SPAN AT TIME 272mL EXPOSED UPON MINING 272m-292m WAS ~15m.SUPPORT IN DWPT IS 2.4m LONG #6 REBAR ON 1.2m X 1.2m PATTERN + SHOTCRETE (2") + WELDMESH. 10-14m LONG CABLES IN BACK ~2.5m X 2m PATTERN (2X CABLE BULGE – 50tonne). NOTE MESH PUT ON AT END.

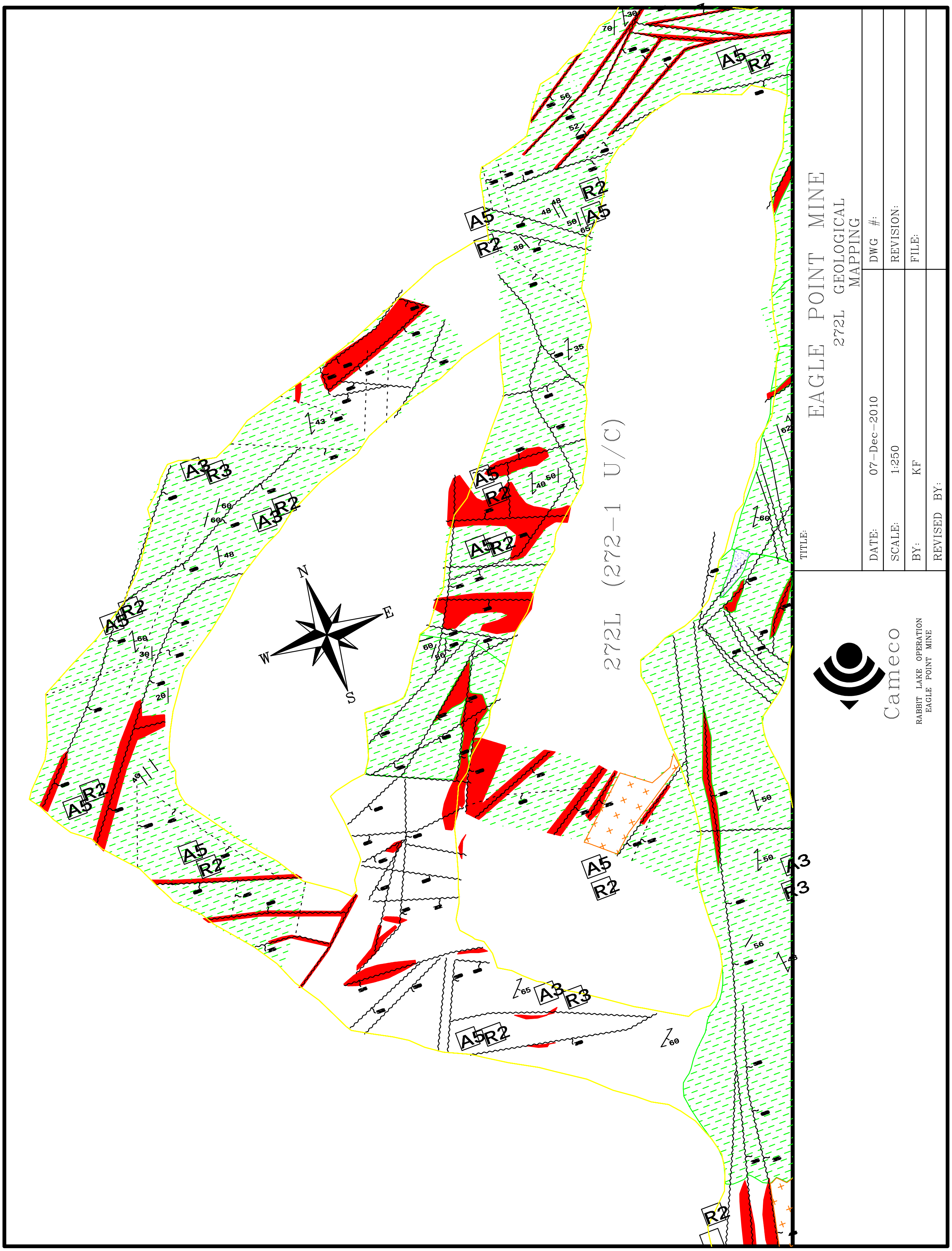


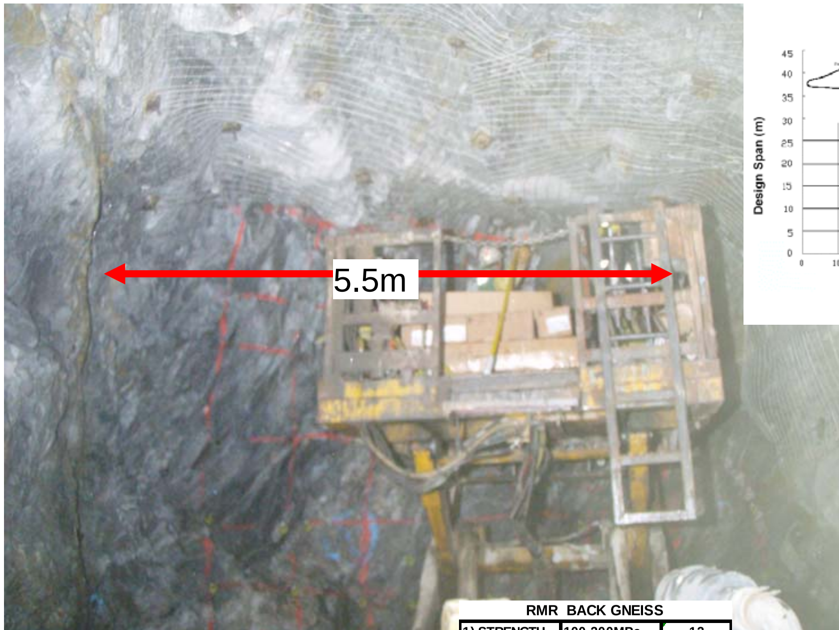
NOTE FW IS VERTICAL AND ACTS AS GEOMETRIC HW.INDICATES SHOULD HAVE ~1m+ ELOS. NOTE FW SLOUGH ON WEDNESDAY MAY 21 RESULTED IN BURRIED SCOOP. (STRUCTURE + ELOS – JR)

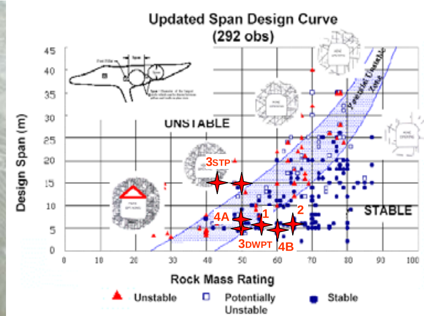


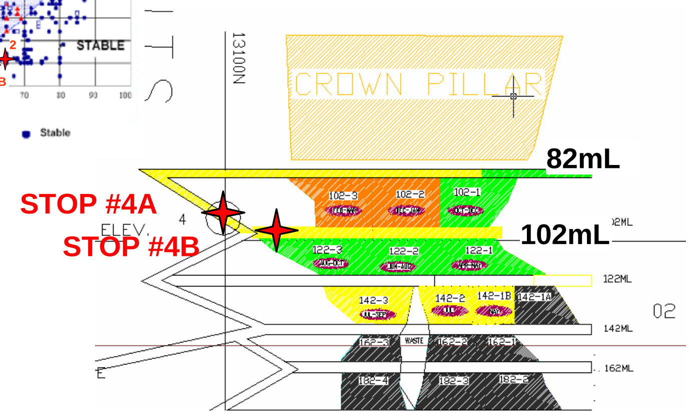
STOP #3: 272mL, 272-1 STOPE U/C (252mL – 272mL) 02 NEXT (02 NORTH EXTENSION)

LOOKING NORTH TOWARDS VOID FROM 272mL TO 252mL OF STOPE 272-1 STOPE. THIS IS THE SECOND LIFT OF THAT MINED FROM 292mL-272mL AND OBSERVED IN DEC'2007 AUDIT. GENERALLY THE SLOPE DISTANCE OF HW WAS 24m X STRIKE LENGTH O F STOPE OF 20m RESULTING IN HR=5.5m. THE RMR OF HW/FW WAS 50% OR Q'=1.9 (AP) WITH INCLINATION OF 62º (C=5.2) WITH X-CUTTING JOINTS (B=0.2) RESULTING IN N'=1.9. NOTE FW WAS VERTICAL HAVING VERTICAL HT FROM UNSUPPORTED FW FROM FLOOR OF 272mL TO FLOOR OF 252mL ~24m HT RESULTING IN HR_{FW}=5.5m AND N_{FW}'=1.9 X 1 X 0.2 X 8 = 3.0. NOTE SHOULD ENSURE HW DISTANCE IS STANDARDIZED FROM MID HT TO MID HT OF U/C AND O/C LEVEL.

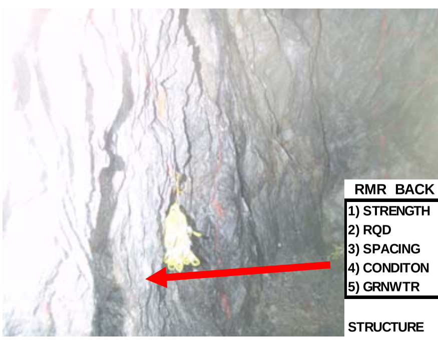

EAGLE POINT MINE
272L GEOLOGICAL MAPPING
TITLE:
DATE: 07-Dec-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
272L (272-1 U/C)
N
E
S
W
A5
R2
A3
R3





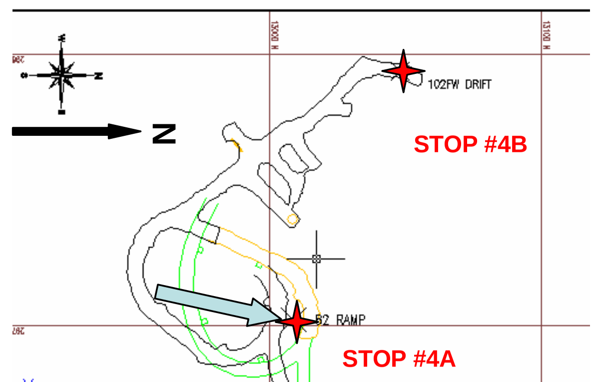


RMR BACK GNEISS		
1) STRENGTH	100-200MPa	12
2) RQD	75%-50%	13
3) SPACING	50-300mm+	15-10
4) CONDITON	TIGHT-SLT	20-12
5) GRNWTR	DRY-MOIST	10-7
	RATING	67%-54%
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	50%



RMR BACK GNEISS/CLAY POCKETS		
1) STRENGTH	25-50MPa+	7-4
2) RQD	50%-25%	8
3) SPACING	50-300mm	10
4) CONDITON	SLT	12
5) GRNWTR	DRY-MOIST	10-7
	RATING	47%-41%
STRUCTURE	FLAT	
	DESIGN (BACK)	45%

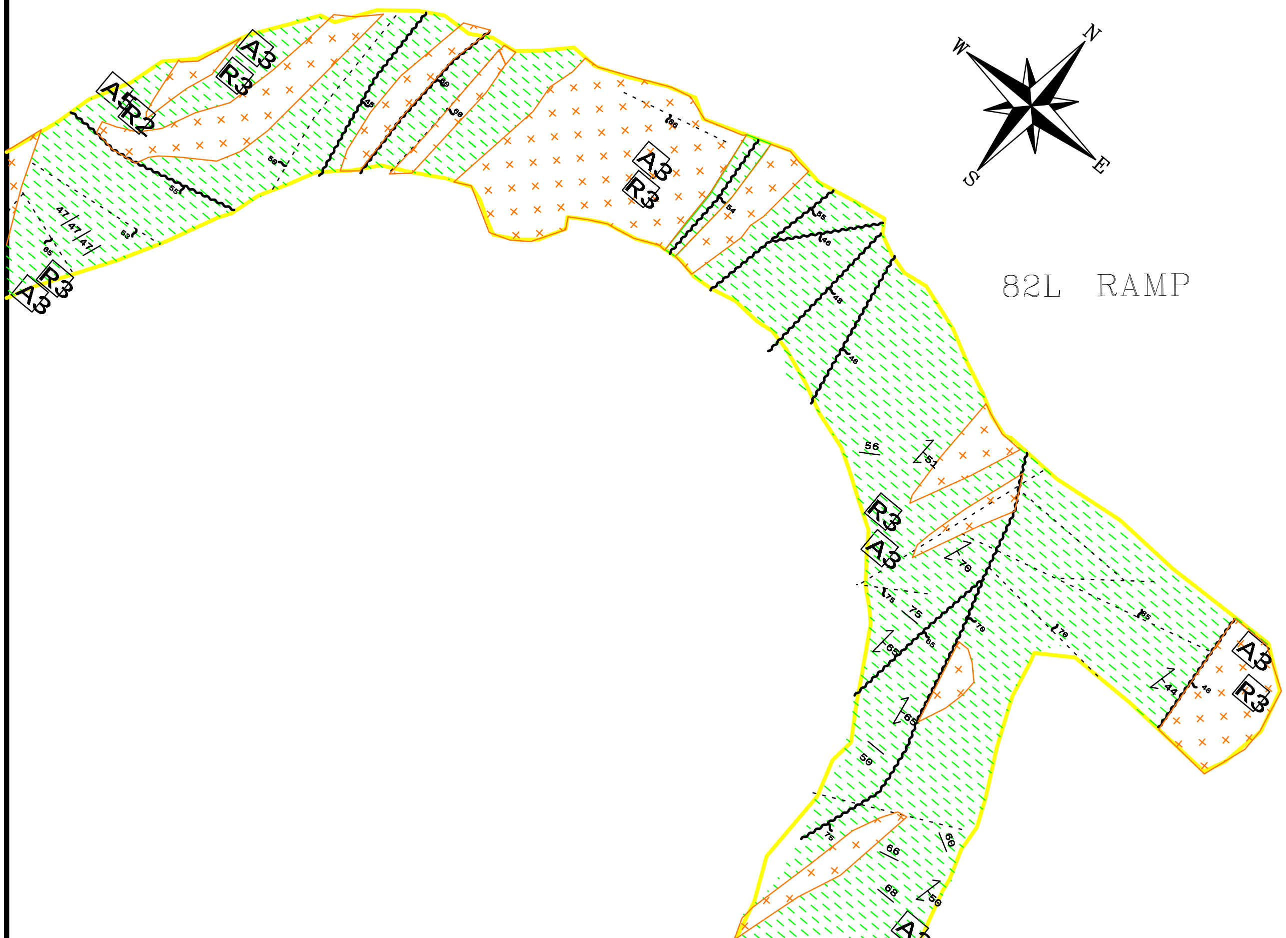
ISOLATED AREAS OF ALTERED WEAK ROCK/CLAY ~<10% OF AREA REST HIGH RMR.



PLAN: 102mL TO 82mL RAMP

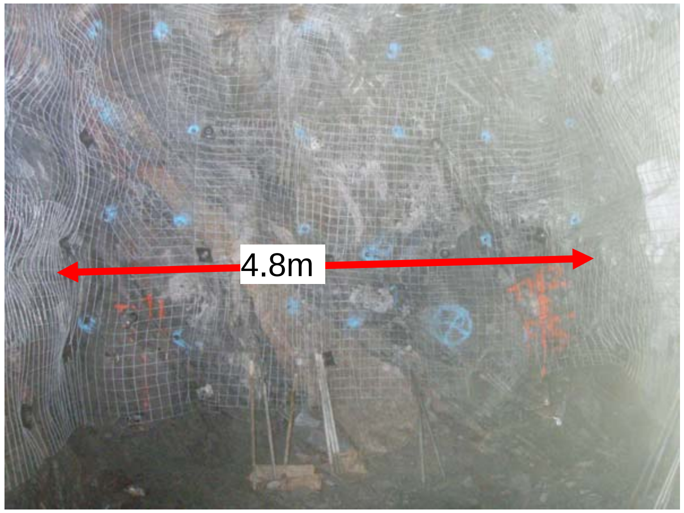
STOP #4A: 82 RAMP, 02 NEXT ACCESS FROM 102mL-82mL

LOOKING EAST AT EXISTING DEVELOPMENT FACE TOWARDS 82mL. SUPPORT COMPRISED OF 2.4m LONG #6 REBAR ON 1.2m X 1.2m PATTERN. + 9GUAGE WWM SCREEN. TYPICALLY IN DEVELOPMENT WASTE WOULD USE MECHANICAL BOLTS, HOWEVER, DUE TO PRESENCE OF CALY AREAS HAS MOVED TOWARDS FULLY BONDED REBAR. SUGGESTED BOND STRENGTH IS 20tons/m (JR) SHOULD BE EVALUATED AND DATABASE KEPT. SCREEN IN BACK AND SHOULDER FOR DEVELOPMENT HEADINGS. DISCUSSION WITH RICK KEOUGH (JUMBO OPERATOR/SHIFTER).. HEIGHT OF RAMP IS 4.9m. NOMINAL 5.2m (W) x 4.9m (H). DESIGN RMR FOR BACK IS 50%+. STABLE WITH SUPPORT DUE TO ISOLATED CLAY/BLOCKY.

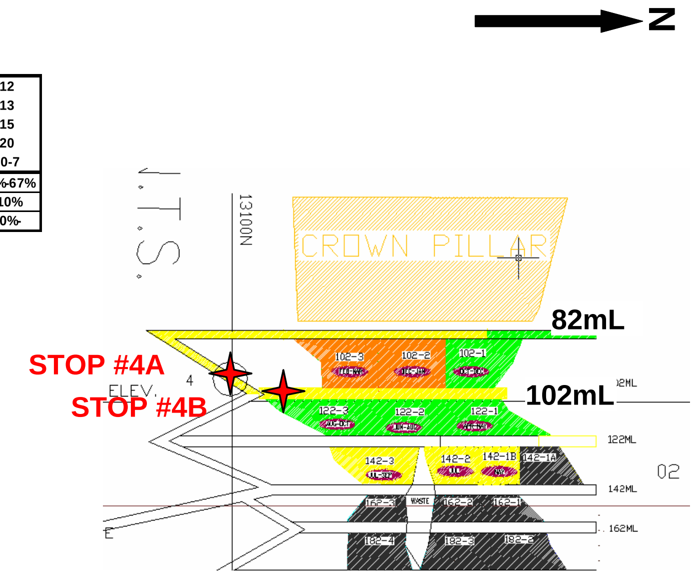


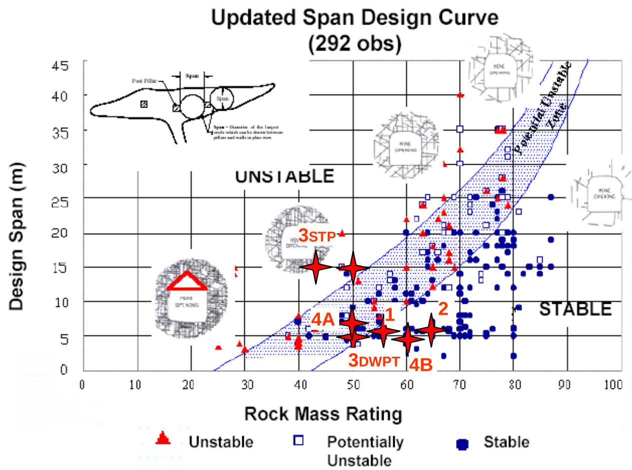
TITLE:	EAGLE POINT MINE 82L GEOLOGICAL MAPPING
DATE: 06-Dec-2010	DWG #:
SCALE: 1:250	REVISION:
BY: KF	FILE:
REVISED BY:	

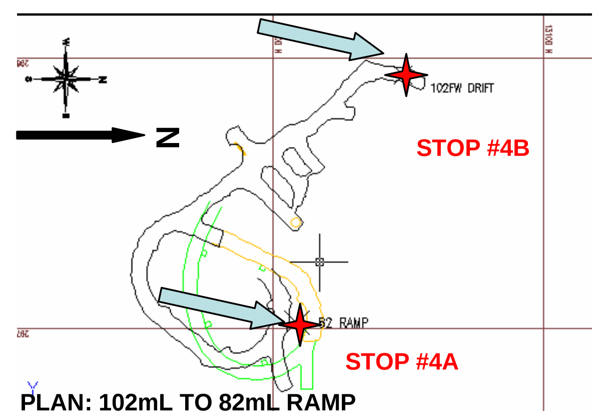
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE



RMR BACK/WALL GNEISS		
1) STRENGTH	100-200MPa	12
2) RQD	75%-50%	13
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT	20
5) GRNWTR	DRY-MOIST	10-7
	RATING	70%-67%
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	60%

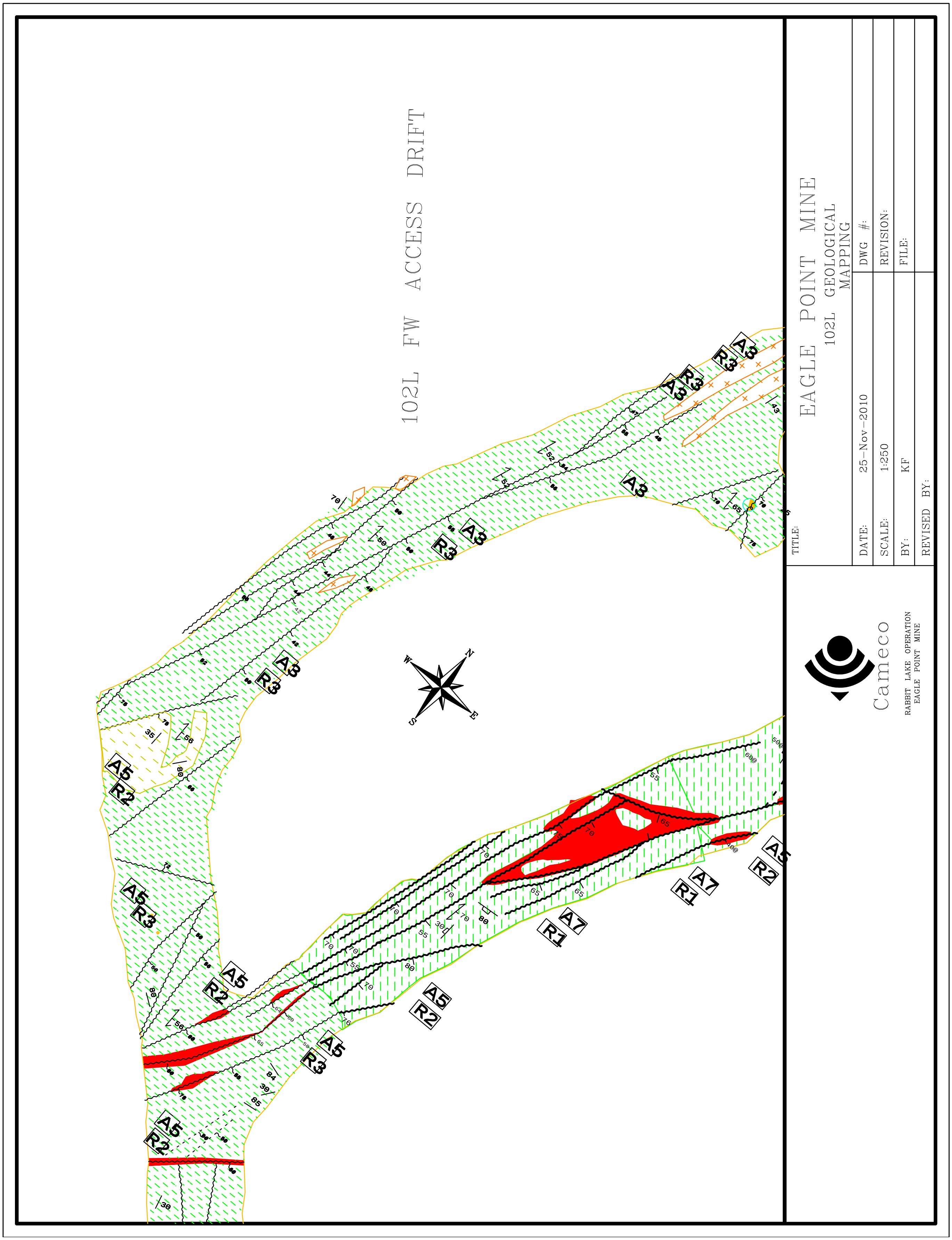




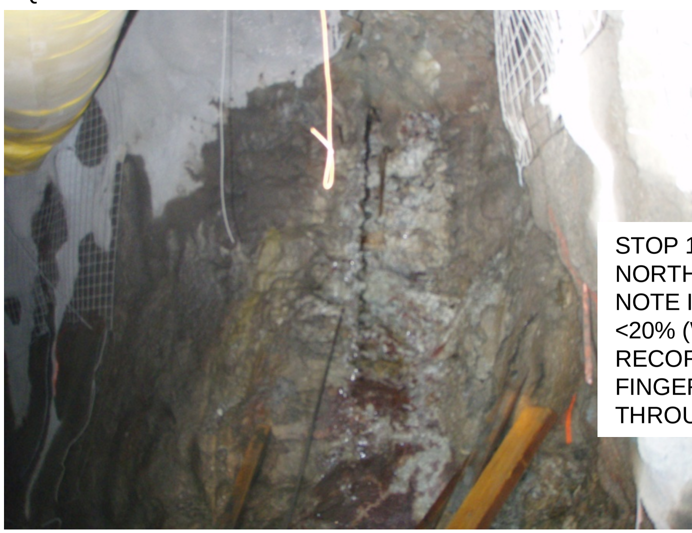


STOP #4B: 102mL FW DEVELOPMENT (02 NEXT STOPES)

LOOKING NORTH AT FACE OF 102mL FW DRIVE DEVELOPMENT FOR "02 NEXT" STOPES. TEST HOLES WITH ONLY TRACES OF WATER RECORDED. SPAN IS 4.8m WIDE WITH NOMINAL FW DEVELOPMENT BEING 4.3m (W) X 4.7m (H). SUPPORT COMPRISED OF 2.4m LONG #6 REBAR ON 1.2m X 1.2m PATTERN IN BACK WITH 6ft SPLITSETS ON 1.2m X 1.2m PATTERN IN WALL. SCREEN TO WITHIN 2FT OF FLOOR. DESIGN RMR FOR BACK IS 60% DUE TO FLAT JOINTS. STABLE.


102L FW ACCESS DRIFT
TITLE:
EAGLE POINT MINE
102L GEOLOGICAL MAPPING
DATE: 25-Nov-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE

STOP 1A: NORTH WALL AREA WAS SHOTCRETED THOROUGH THE PERMEABLE QUARTZ ZONE. WET FLOWING ~30-40GPM

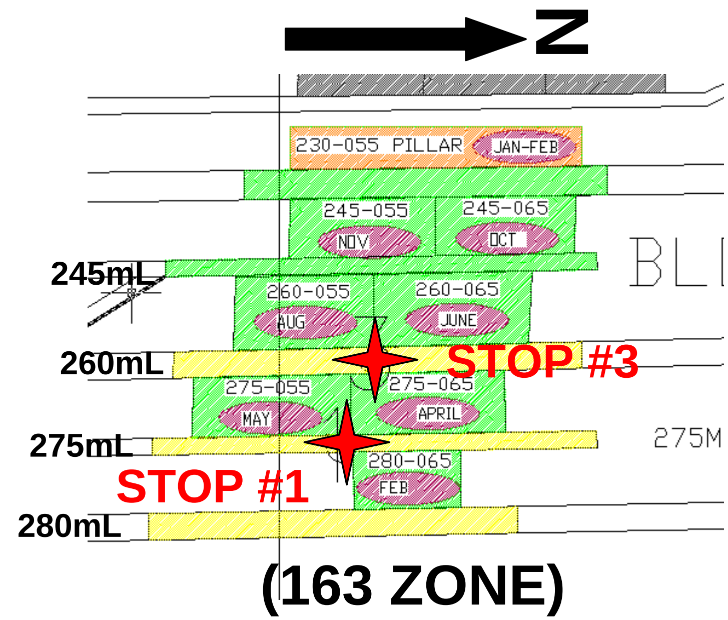


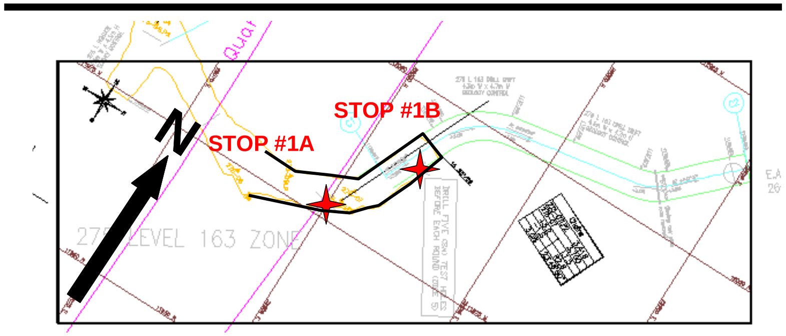
STOP 1B: FACE LOOKING NORTH. RMR WAS 30%. NOTE IF WET COULD BE <20% (WATER/CLAY). CG RECORDED <15% - FINGERS ABLE TO PUSH THROUGH.

RMR WALL/BACK/FACE (ALTERED)

1) STRENGTH	25-50MPa	4
2) RQD	50%-25%	8-3
3) SPACING	50mm+	10-5
4) CONDITON	GOUGE	6
5) GRNWTR	DRY	10*
	RATING	38-28
STRUCTURE		
	DESIGN (BACK)	30%

*** DRY NOW, HOWEVER POTENTIALLY WATER WOULD BE MOD TO HIGH PRESSURE ie 30-40GPM+ OR DRY WEIGHTING OF "4-0"**



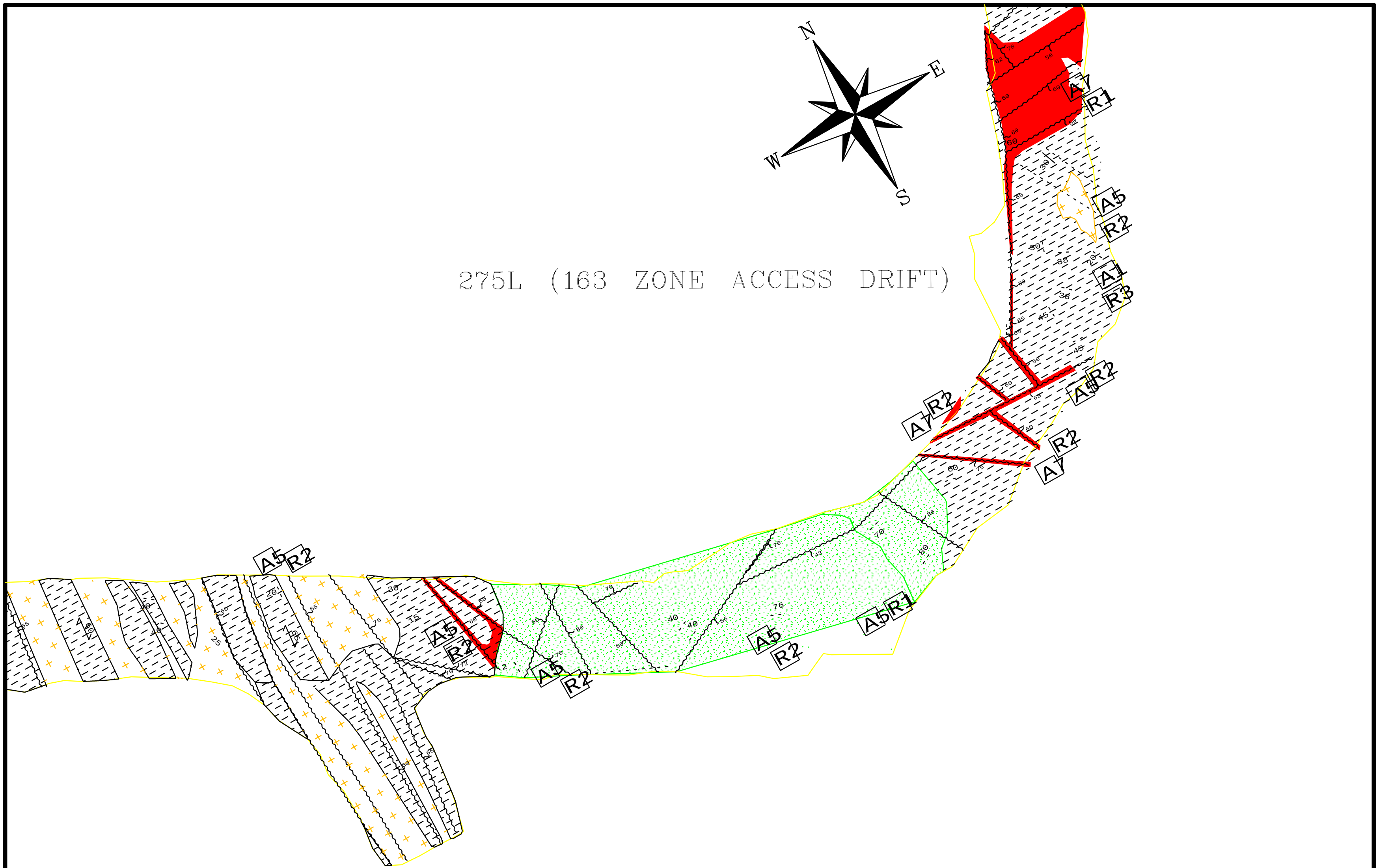


275mL PLAN

STOP #1: 275L, 163 DRILL DRIFT (275-260 163 ZONE)

CLASSIFIED AS A7/R1 (STRONGLY ALTERED AND VERY WEAK ROCK) BY GEOLOGY AND THIS WOULD BE Q'=0.4 ~ RMR=36%). WALLS/BACK STABILIZED WITH 10cm SHOTCRETE, PATTERN BOLTING OF 2.4m LONG REBAR ON 1.2m X 1.2m PATTERN + 9GAUGE SCREEN ON BACK AND WALL WITH A FURTHER 10cm OF SHOTCETE OVER THE PLACED SUPPORT. AREA SHOWS NO SIGNS OF CRACKING. SPAN IS 4.4m (W) X 4.3m (H). PRESENT PATTERN IN PROXIMITY OF FACE AS GROUND HAS IMPROVED (DRY) IS EMPLOYING 5cm OF SHOTCRETE DUE TO RADON WITH STD #6 REBAR PATTERN IN BACK (2.4m LONG ON 1.2m X 1.2m PATTERN+ SCREEN) WITH 1.8m SPLITSET AT BOTTOM ROW ~ 1m FROM FLOOR. IMPORTANT TO ENSURE DOES NOT UNDERCUT SHOTCRETE ABOVE.

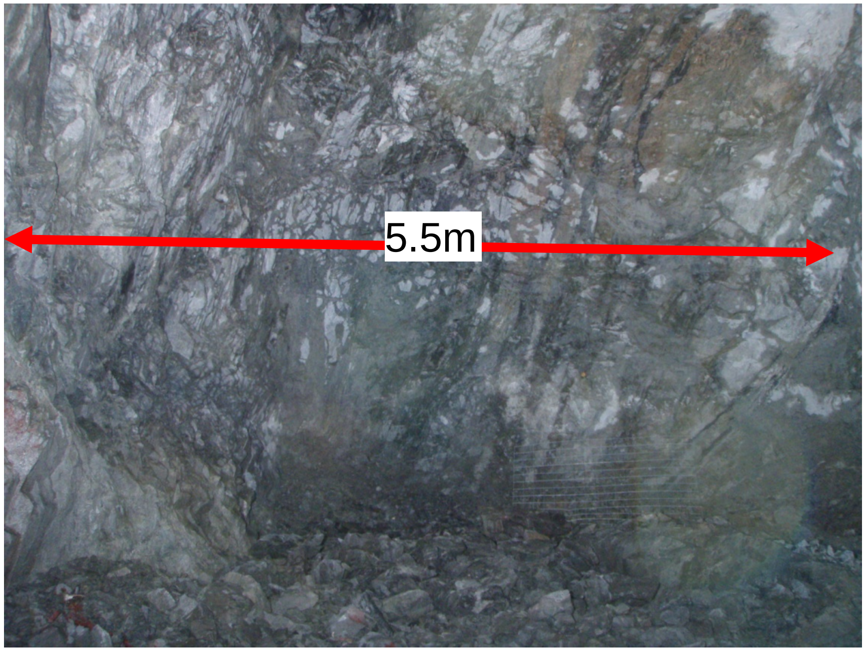
5

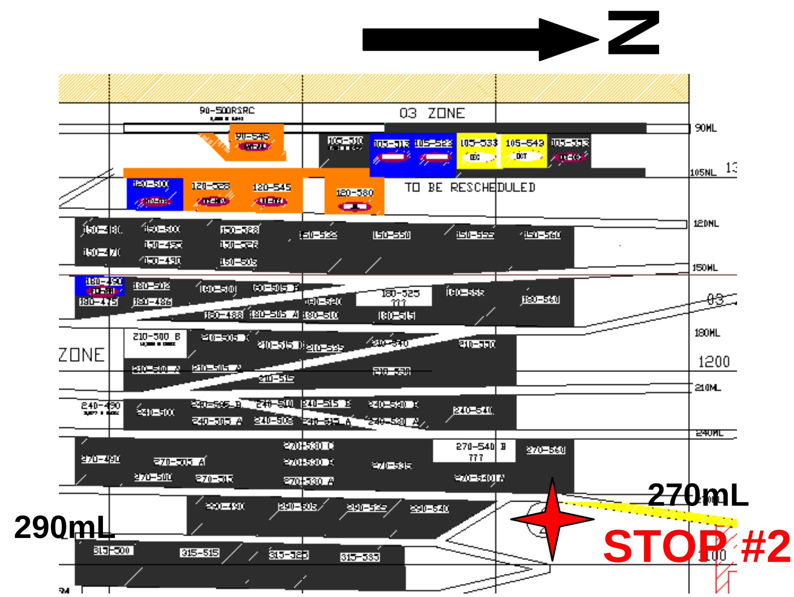


Cameco

RABBIT LAKE OPERATION
EAGLE POINT MINE

TITLE:	EAGLE POINT MINE 275L GEOLOGICAL MAPPING		
DATE:	25-Nov-2010	DWG #:	
SCALE:	1:250	REVISION:	
BY:	KF	FILE:	
REVISED BY:			



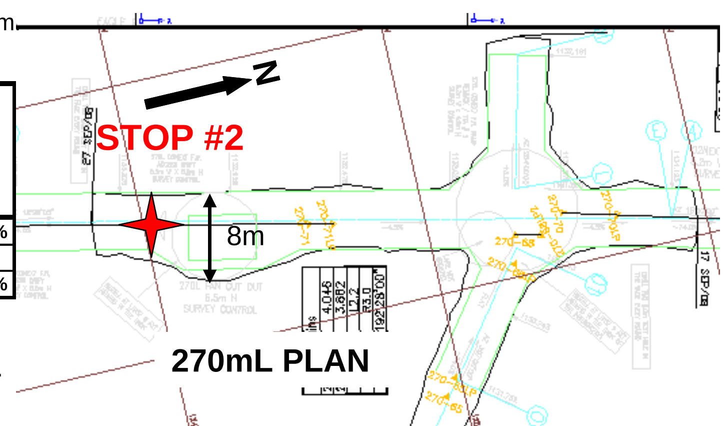


LOOKING AT FACE. SPAN IS 5.5m.

RMR WALL/BACK/FACE (GNEISS)

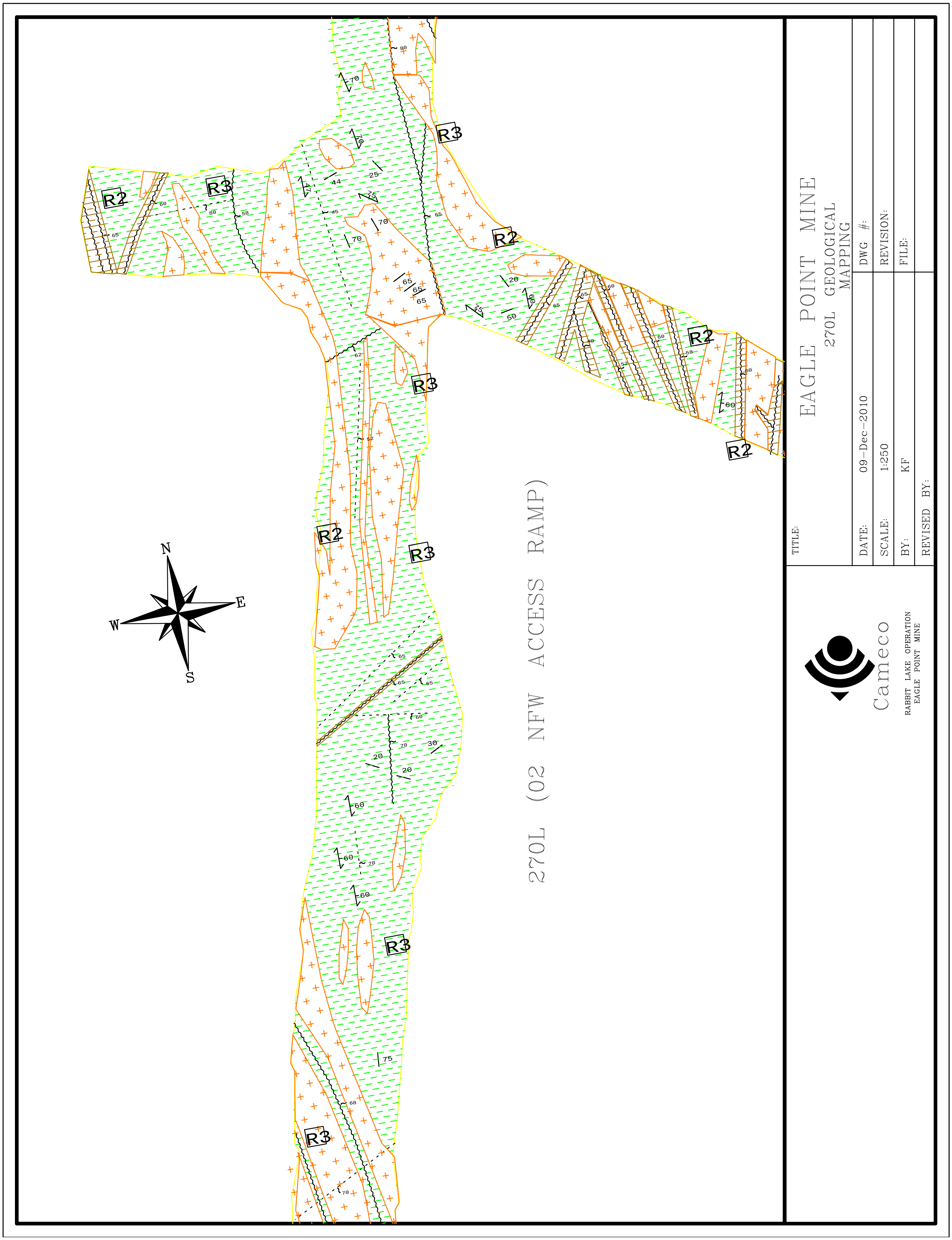
1) STRENGTH	100-200MPa	12
2) RQD	75%-50%	13
3) SPACING	300mm+	10
4) CONDITON	TIGHT-SLT	20-12
5) GRNWTR	WET	10
	RATING	65%-57%
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	50%-55%

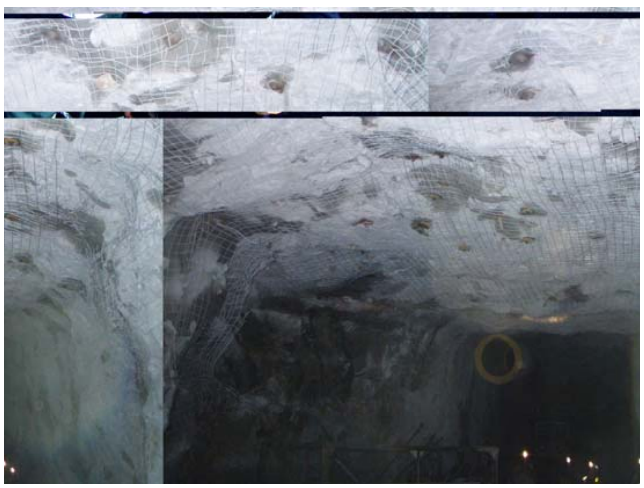
LOOKING AT BACK. SPAN IS 8m.



STOP #2: 270L EXHAUST ACCESS DRIFT

THIS IS A LONG TERM DRIFT ACCESS/MAIN FOR HAULAGE FOR VENTILATION AND SUPPORT IN BACK IS 2.4m LONG #6 REBAR IN BACK ON 1.2m X 1.2m PATTERN WITH 1.8m LONG SPLITSETS IN WALL ON 1.2m X 1.2m PATTERN. WELDMESH(9 GAUGE) ALONG BACK AND 1.2m DOWN FROM SHOULDERS. RMR IN BACK IS 50%+ WITH DRIFT SPAN GENERALLY UNDER 5.5m AT FACE WITH ISOLATED AREA IN PROXIMITY OF FAN CUTOUT OF 8m. THE 8m SPAN IS POTENTIALLY UNSTABLE FOR THE 50%+ RMR. EXTENSION STEEL EMPLOYED AND TREATED AS INTERSECTION (16ft REBAR).

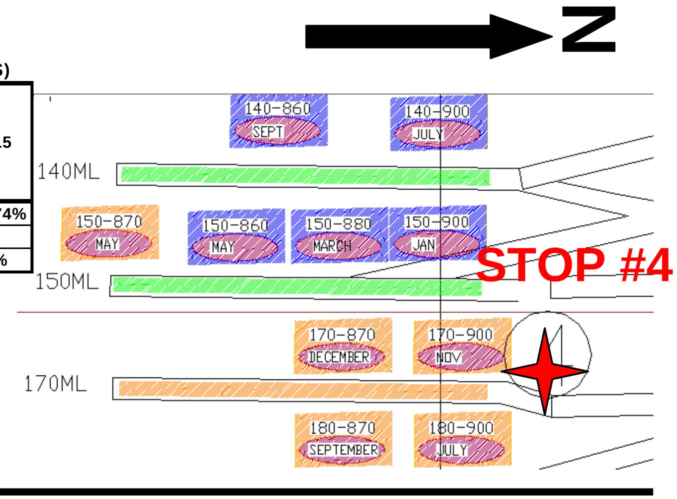

270L (02 NFW ACCESS RAMP)
N
E
S
W
R2
R3
TITLE:
EAGLE POINT MINE
270L GEOLOGICAL MAPPING
DATE: 09-Dec-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE

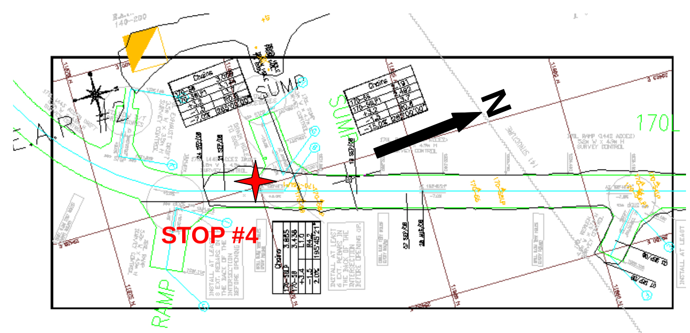


FACE

RMR WALL/BACK/FACE (GNEISS)

1) STRENGTH	100-200MPa	12
2) RQD	90%-75%	17
3) SPACING	0.3-1m-	20-15
4) CONDITON	TIGHT	20
5) GRNWTR	WET	7
	RATING	79%-74%
STRUCTURE	FLAT (NONE)	
	DESIGN (BACK)	75%

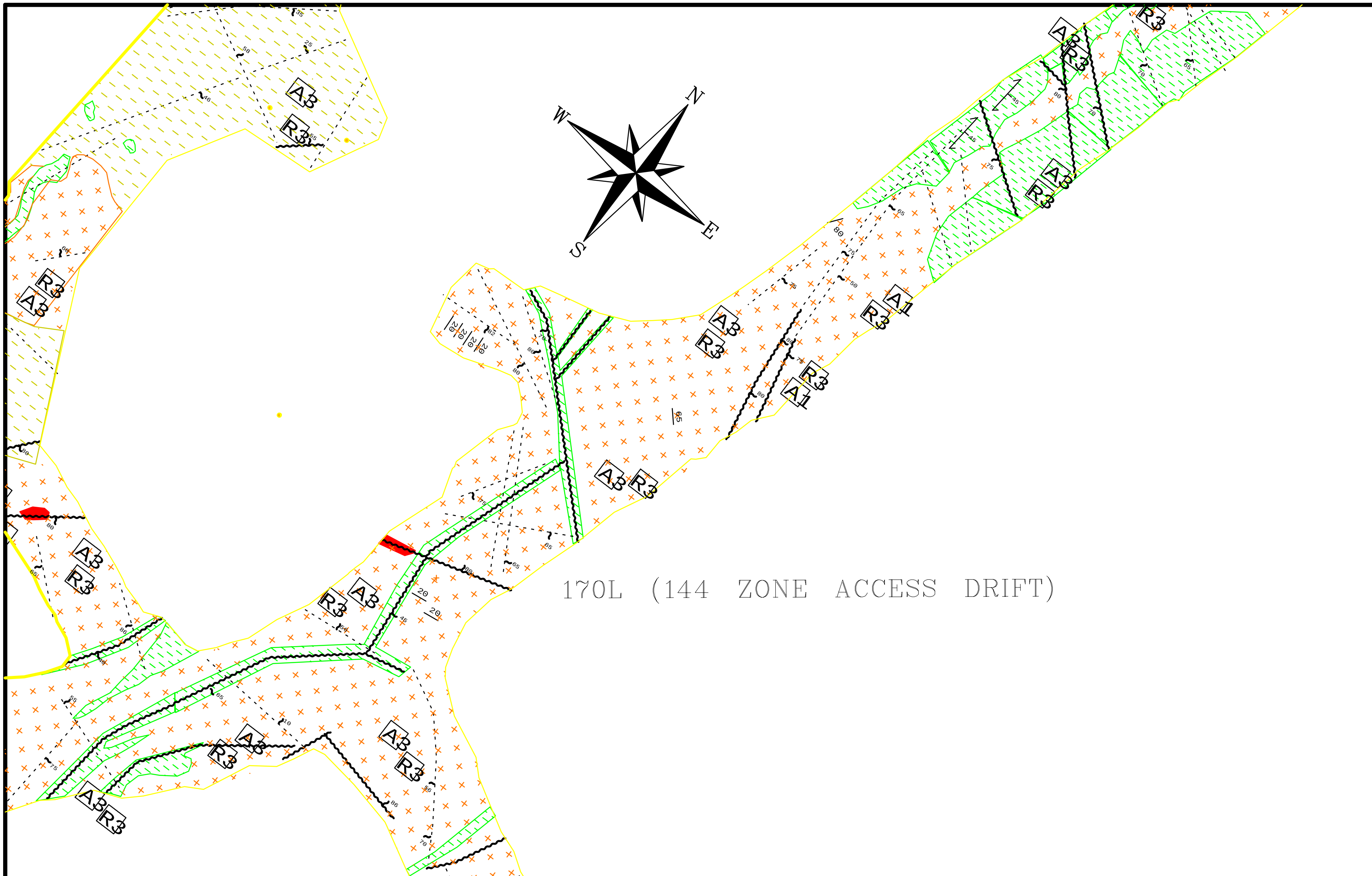




170 LEVEL PLAN

STOP #4: 170L RAMP DEVELOPMENT (WASTE) – 144 ZONE SOUTH

THIS IS AN INTERSECTION AND THEREFORE DESIGNED WITH EXTENSTION STEEL FOR SUPPORT(STABLE) .170L DEVELOPMENT IN WASTE WITHIN INTERESECTION APPROACHING 8m WITH 75% RMR IS STABLE, STRUCTURE INDICATES STABLE, HOWEVER, DUE TO SPAN 16ft EXTENSTION #6 REBAR EMPLOYED ON 1.2m X 1.2m PATTERN. SPLITS IN BACK WITH 2.4m LONG MECHANICALS IN GOOD GROUND ON 1.2m X 1.2m PATTERN WITH 1.8m LONG SPLITSETS IN WALL ON 1.2m x 1.2m PATTERN + SCREEN FROM BACK TO 1.2m DOWN SHOULDERS. HAVE TWO(2) SQUARE OVERLAP.



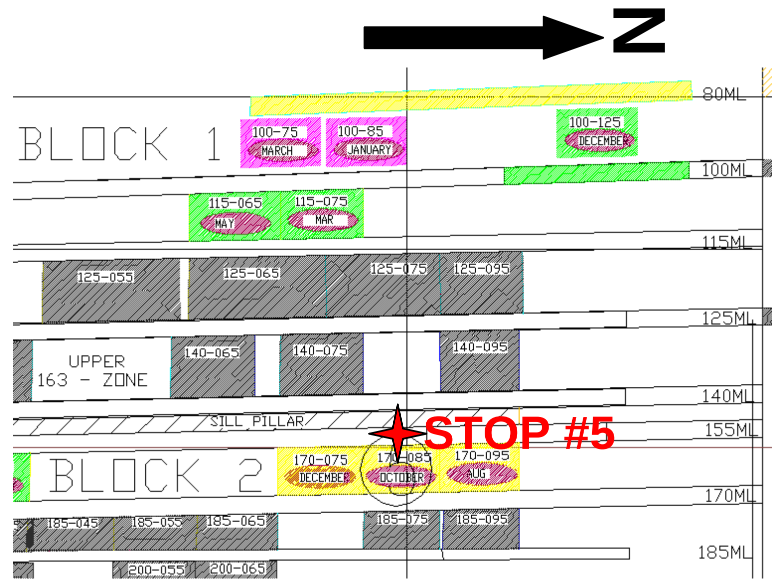
TITLE: EAGLE POINT MINE 170L GEOLOGICAL MAPPING	
DATE: 25-Nov-2010	DWG #:
SCALE: 1:250	REVISION:
BY: KF	FILE:
REVISED BY:	

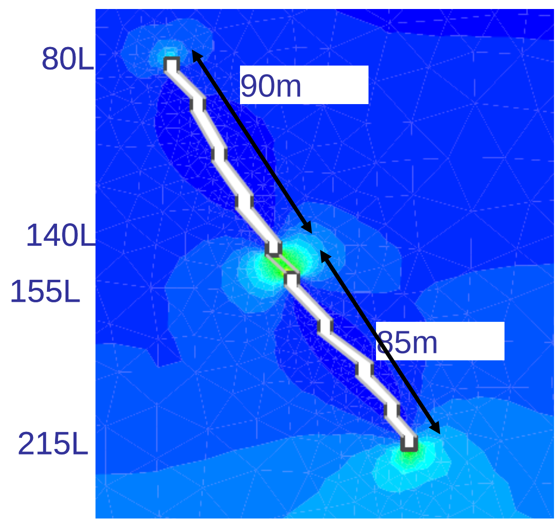
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE

LOOKING NE TOWARDS 170-095 STOPE FILLED WITH CRF/URF FROM 155-170 LEVELS.

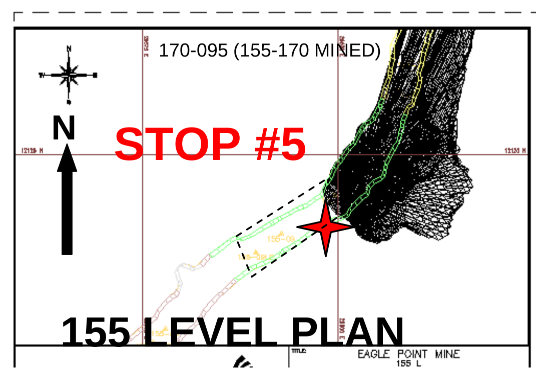
RMR WALL/FW (GNEISS)

1) STRENGTH	100-200MPa	12
2) RQD	75%-50%	13
3) SPACING	50-300mm	10
4) CONDITON	TIGHT-SLT	20-12
5) GRNWTR	DRY	10
	RATING	65%-57%
STRUCTURE		
	DESIGN (FW)	60%



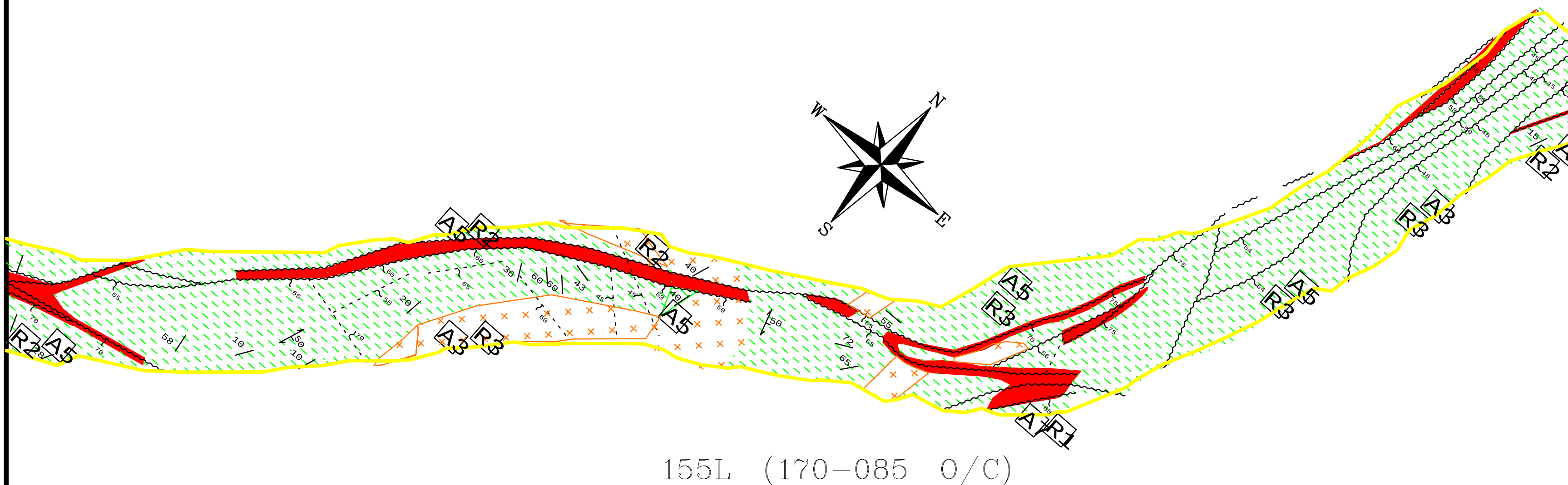


6m+ VERTICAL AND 8m AND OPENING ~5m THEREFORE Wp/Wo= 8/5=1.6



STOP #5: 155L, 170-085 STOPE (U/C) – 163 ZONE

SPAN IS 4.7m HAVING AN RMR IN WALL/BACK ~60%. AREA SHOTCRETED. LOOKING NE TOWARDS MINED OUT STOPE 170-095 STOPE. SILL ABOVE ~6m VERTICAL THICKNESS ABOVE TO 150L. STRIKE LENGTH OF STOPE TO BE MINED IS 23m WITH SLOPE HEIGHT OF 23m. CONCERN IS THE SLOPE OF THE STOPE IS SHALLOW AT 50o . THIS RESULTS IN POTENTIALLY HIGH SLOUGH COUPLED WITH STRUCTURE. RMR IS GENERALLY HIGH (60%), HOWEVER, WITH HR=5.8m AND N=9 (11 X1 X 0.2 X 4.1) RESULTS IN >2m OF WALL SLOUGH. **NOTE GROUND CONTROL LETTER MISSING.**

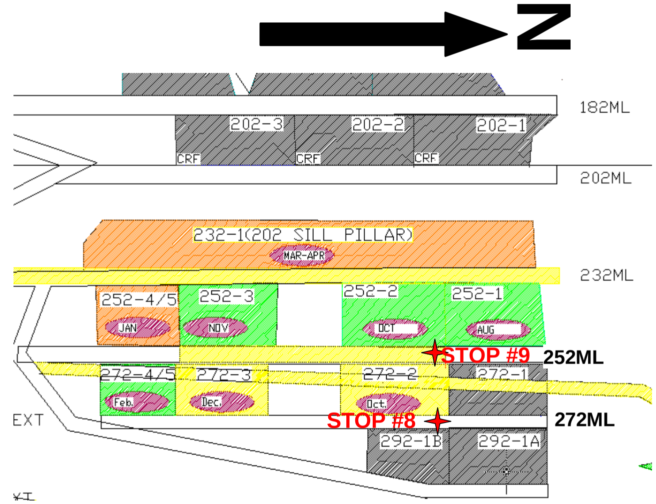


TITLE: EAGLE POINT MINE 155L GEOLOGICAL MAPPING			
DATE:	09-Dec-2010	DWG #:	
SCALE:	1:250	REVISION:	
BY:	KF	FILE:	
REVISED BY:			

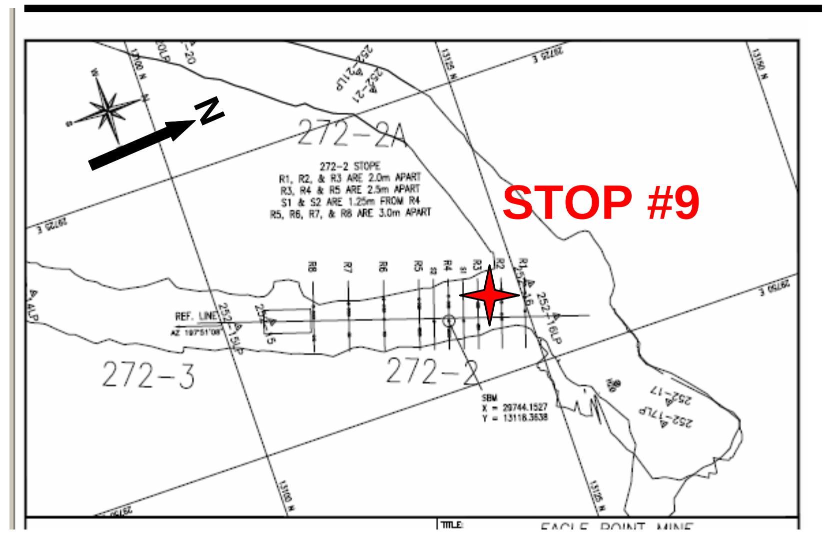
RMR WALL/BK (GNEISS)

1) STRENGTH	100-200MPa	12
2) RQD	75%-50%	13
3) SPACING	50-300mm+	15
4) CONDITON	SLT	12
5) GRNWTR	MOIST/DRIP	7
	RATING	59%
STRUCTURE		
	DESIGN (HW/BK	59%

LOOKING AT END OF STOPE ON 252L. SPAN IS 10m BLOCKY RMR IS 59%, HOWEVER, STRUCTURED. AREA SUPPORTED WITH 16ft #6 REBAR ON 1.2m X 1.2m PATTERN. LOOKING NORTH.



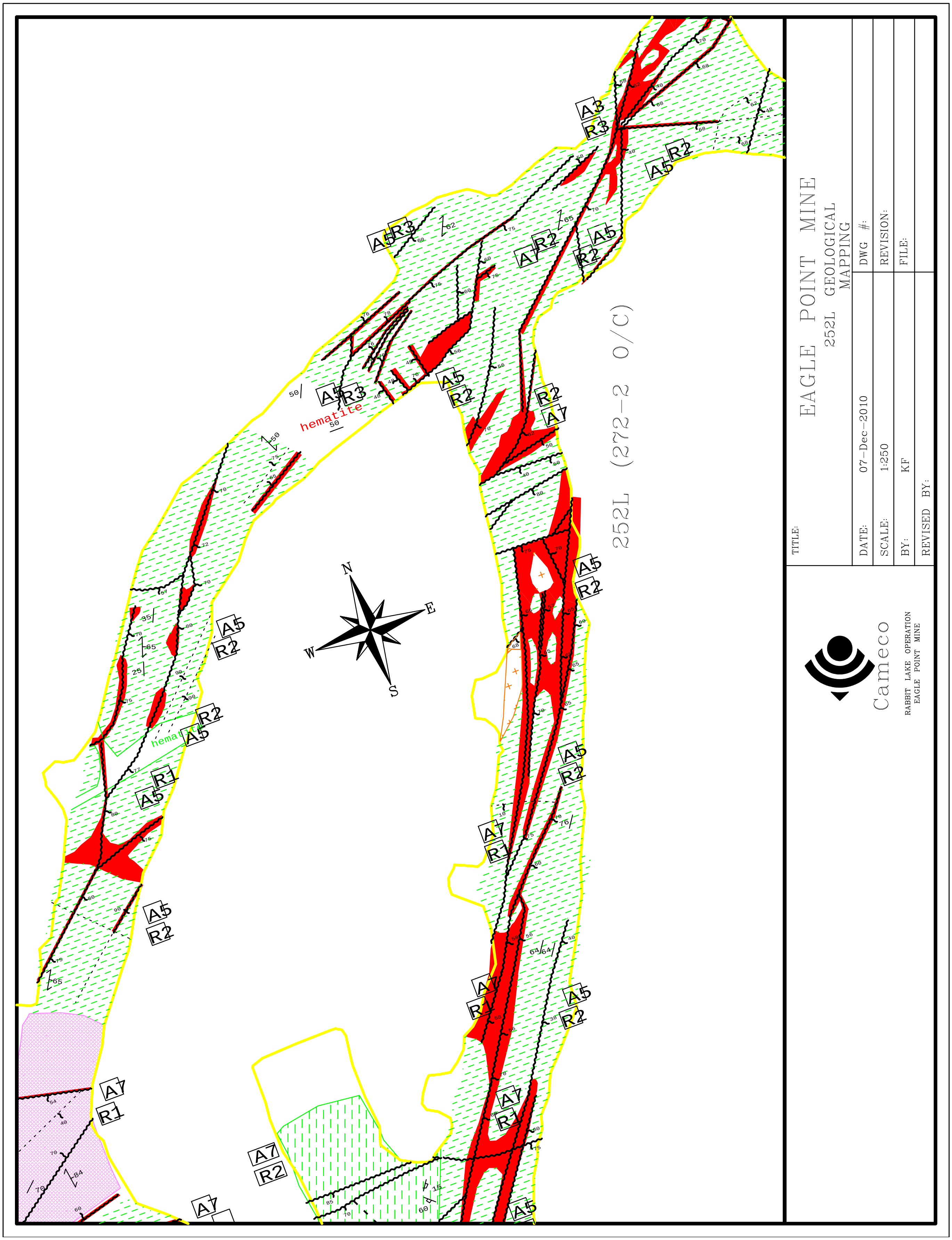
LOOKING AT BACK OF ACCESS AT TOP OF REAMER. SPAN IS 4.7m. LOOKING STH.



252 LEVEL PLAN

STOP #9: 252L, 272-2 STOPE (02 NEXT LOWER ZONE)

LOOKING AT TOP OF 272-2 STOPE 0/C. THIS STOPE IS DESIGNED TO BE 19m IN LENGTH X 25m IN SLOPE HEIGHT (CENTER TO CENTER). THE ESTIMATED ELOS IS <1m GIVEN THE RMR DESIGN IS 59% (GEOLOGY). THE HW IS CABLED.


TITLE:
EAGLE POINT MINE
252L GEOLOGICAL
MAPPING
DWG #:
REVISION:
FILE:
DATE: 07-Dec-2010
SCALE: 1:250
BY: KF
REVISED BY:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
252L (272-2 O/C)
hematite
N
E
S
W

LOOKING AT STOPE. MINIMAL CONCERN. DIP IS SHALLOW ~50º. 18m-20m STRIKE X 16-15m IN HW EXPOSURE (CENTER TO CENTER OF HW).

RMR BACK/WALL GNEISS

1) STRENGTH	100-200MPa	12
2) RQD	90%-75%	17
3) SPACING	50-300mm+	15
4) CONDITON	SLT	12
5) GRNWTR	DRY	10
	RATING	66%
STRUCTURE	NO FLAT	
	DESIGN (BACK)	66%+

163 ZONE

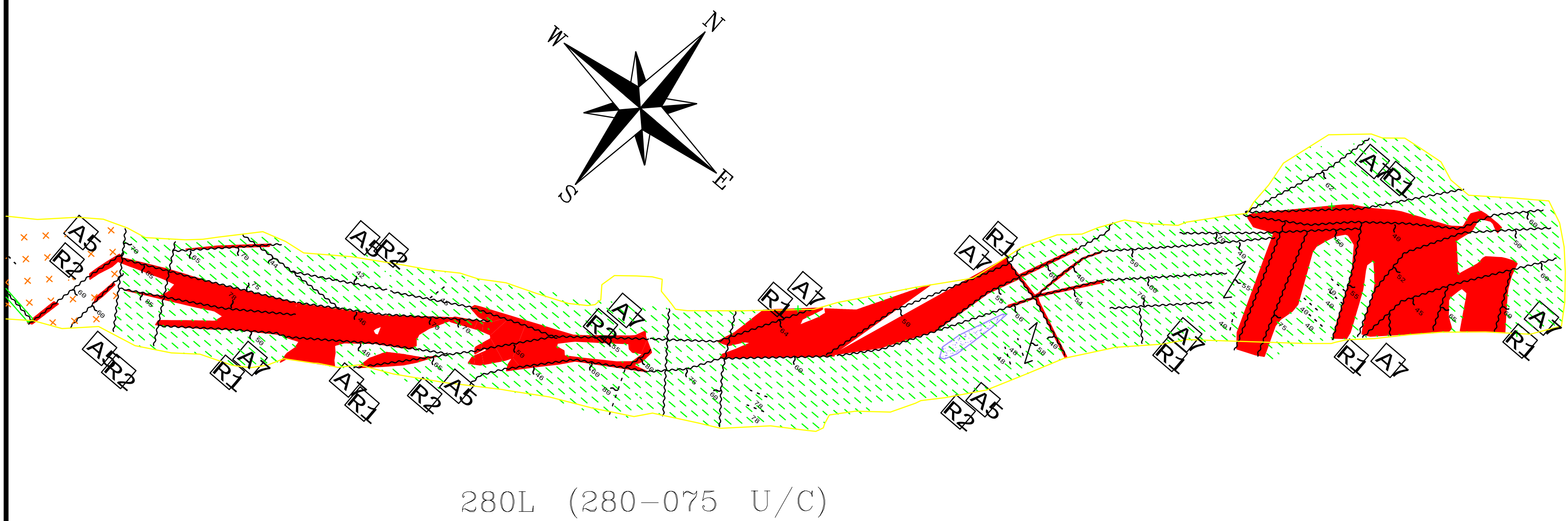
FW EXPOSURE SIMILAR TO HW/BACK. RMR ~65%

SHOTCRETED ACCESS ~50mm SHOTCRETE WITH BOLTS THROUGH THE SHOTCRETE. NOTE NO CRACKS

280L PLAN

STOP #1: 280L, 280-075 STOPE (163 ZONE) STOPE BETWEEN 280L AND 275L

LOOKING(JUNE 1/09) AT STOPE RMR DESIGN IS 60%-65% WITH BACK SPAN OF 4.5m. REBAR + SHOTCRETE IN DWPT AREA. HW IS CABLED. ACCESS SPAN IS 4.5m(W) X 4.7m(H).. PARALLEL JOINTS IN HW OF STOPE. DRAWPOINTS SHOW NO/MINIMAL CRACKS (~50mm THICK). NOTE 12m CABLES INSTALLED ON HW O/C AND HAD ~6m FAILED FROM BACK DUE TO BLAST STRIPPED STD SUPPORT IN DRAWPOINT ACCESS COMPRISED OF #6 - 2.4m LONG REBAR ON 1.2m X 1.2m PATERN + SCREEN WITH SPLITS IN WALL(6ft) SAME PATTERN + 50mm SHOTCRETE. OBSERVED ON JUNE 3/09 AFTER BLAST SIMILAR CONDITIONS. NOTE THIS AREA IS BELOW THE 275L DRILL DRIFT ACCESS TO 163 ZONE THAT REQUIRED INCREASED SUPPORT/GROUND COLLAPSE (SEPT 08 VISIT/EPM-9/08 STOP #1)



Cameco

RABBIT LAKE OPERATION
EAGLE POINT MINE

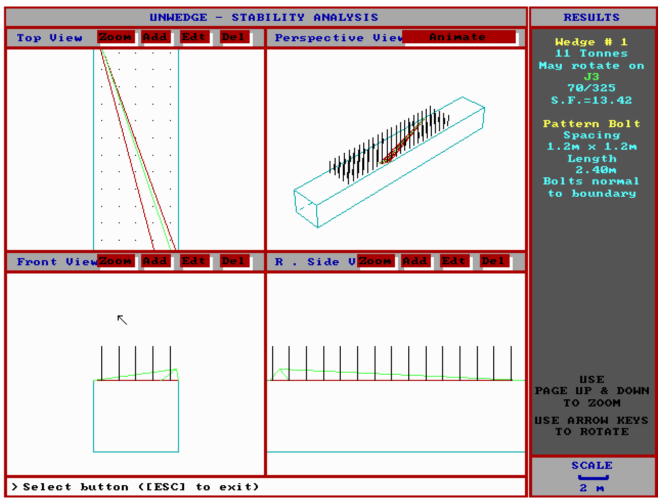
TITLE: EAGLE POINT MINE 280L GEOLOGICAL MAPPING			
DATE:	25-Nov-2010	DWG #:	
SCALE:	1:250	REVISION:	
BY:	KF	FILE:	
REVISED BY:			

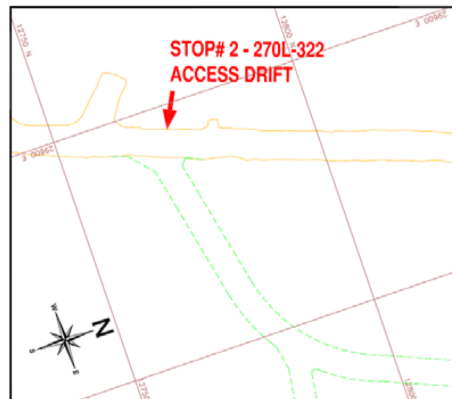
RMR BACK/WALL GNEISS

1) STRENGTH	100-200MPa	12
2) RQD	90%-75%	17
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT	20
5) GRNWTR	DRY	10
	RATING	74%
STRUCTURE	NO FLAT	
	DESIGN (BACK)	74%+

JS	DIP	DDR
1	35^{o}	270^{o}
2	66^{o}	085^{o}
3	70^{o}	325^{o}

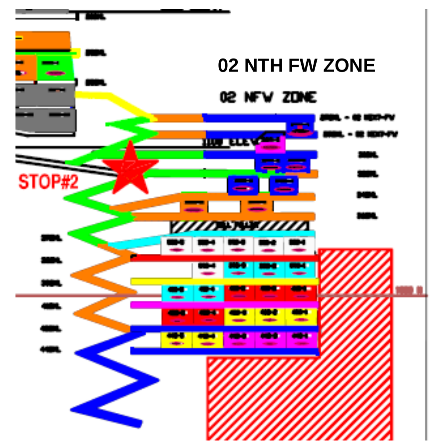
WALL TREND=015º (NOTE CORRECTED TO MINE GRID -025º)





PLAN 270-322L 02 NTH FW ACCESS

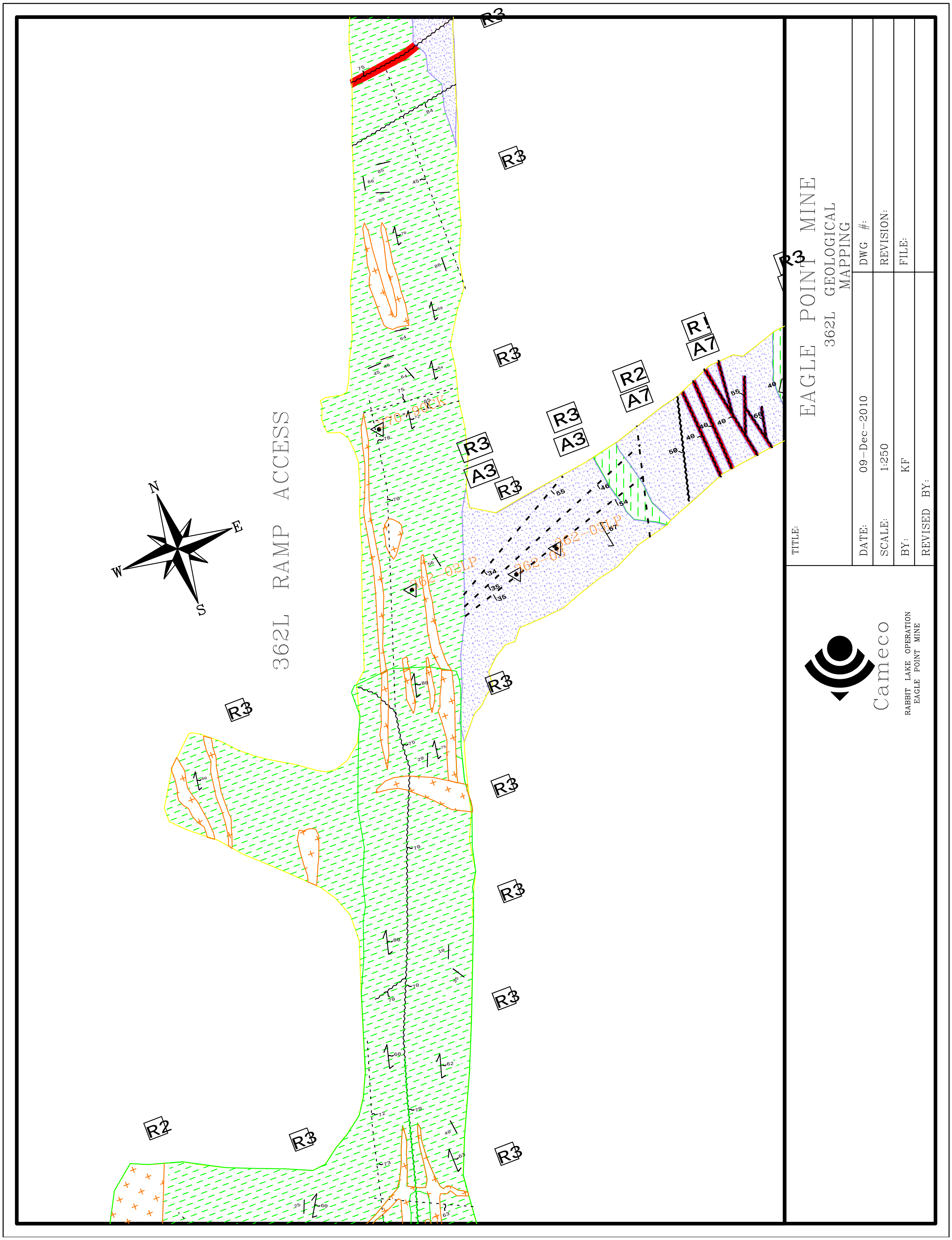
SUPPORT COMPRISED OF 2.4m LONG #6 REBAR BREAKING STRENGTH OF 18tonne AND BOND STRENGTH OF 12t/m (WEAK ROCK MASS). SG=2.7. APEX DEPTH <1m. 11tonne WEDGE. SUPPORTED FS>10.0



STOP #2: 270-322 ACCESS DRIFT FW DEVELOPMENT RAMP TO 02 NEXT FW ZONE

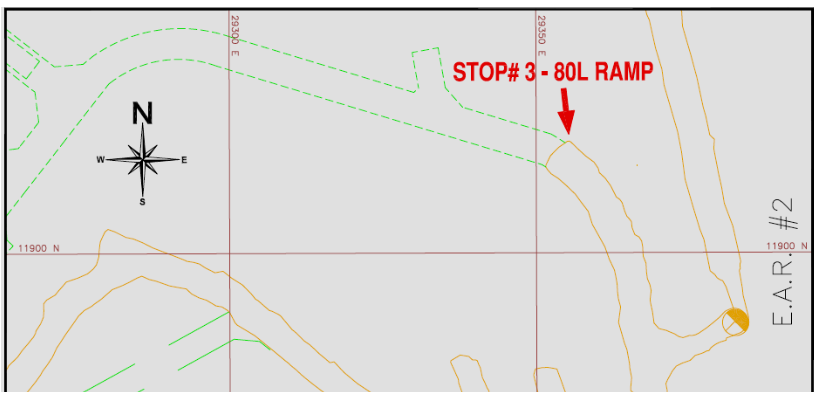
LOOKING AT ACCESS DRIFT FW RAMP DEVELOPMENT TO “02 NEXT” FW ZONE. THE SPAN IS 6m X 7m WITH INTERSECTION WITH EXTENSION REBAR (16ft). THIS AREA WILL BE AN INTERSECTION HAS 74%+ RMR. INTERSECTION IF >8m HAVE XTENSION REBAR INSTALLED. MAPPING AT STN #87. EMPLOYING 2.4m LONG MECHANICAL IN BACK ON 2.4m X 2.4m PATTERN + SCREEN WITH 6ft SPLIT SETS IN WALL ON SAME PATTERN NO FLAT JOINTS.

9


362L RAMP ACCESS
N
E
S
W
R3
A3
R2
A7
TITLE:
EAGLE POINT MINE
362L GEOLOGICAL MAPPING
DWG #:
REVISION:
FILE:
DATE: 09-Dec-2010
SCALE: 1:250
BY: KF
REVISED BY:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE

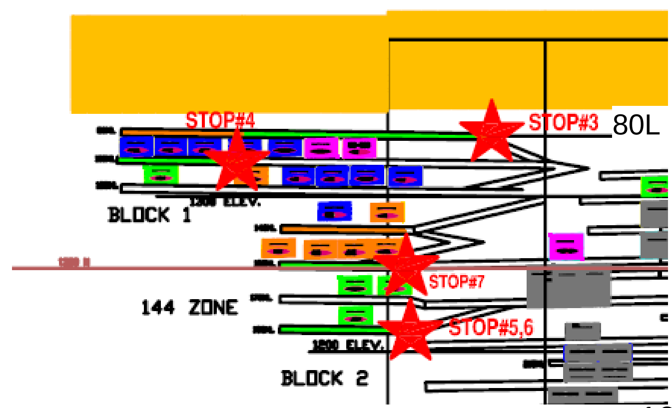
RMR BACK/WALL PEGMATITE		
1) STRENGTH	100-200MPa	12
2) RQD	75%-50%	13
3) SPACING	50-300mm	10
4) CONDITON	TIGHT-SLT	20-12
5) GRNWTR	MOIST	7
STRUCTURE	RATING	72%-54%
	NO FLAT	
	DESIGN (BACK)	60%+

ACCESS DRIFT HIT WATER ~10GPM – STOPPED ADVANCE.



PLAN 80L

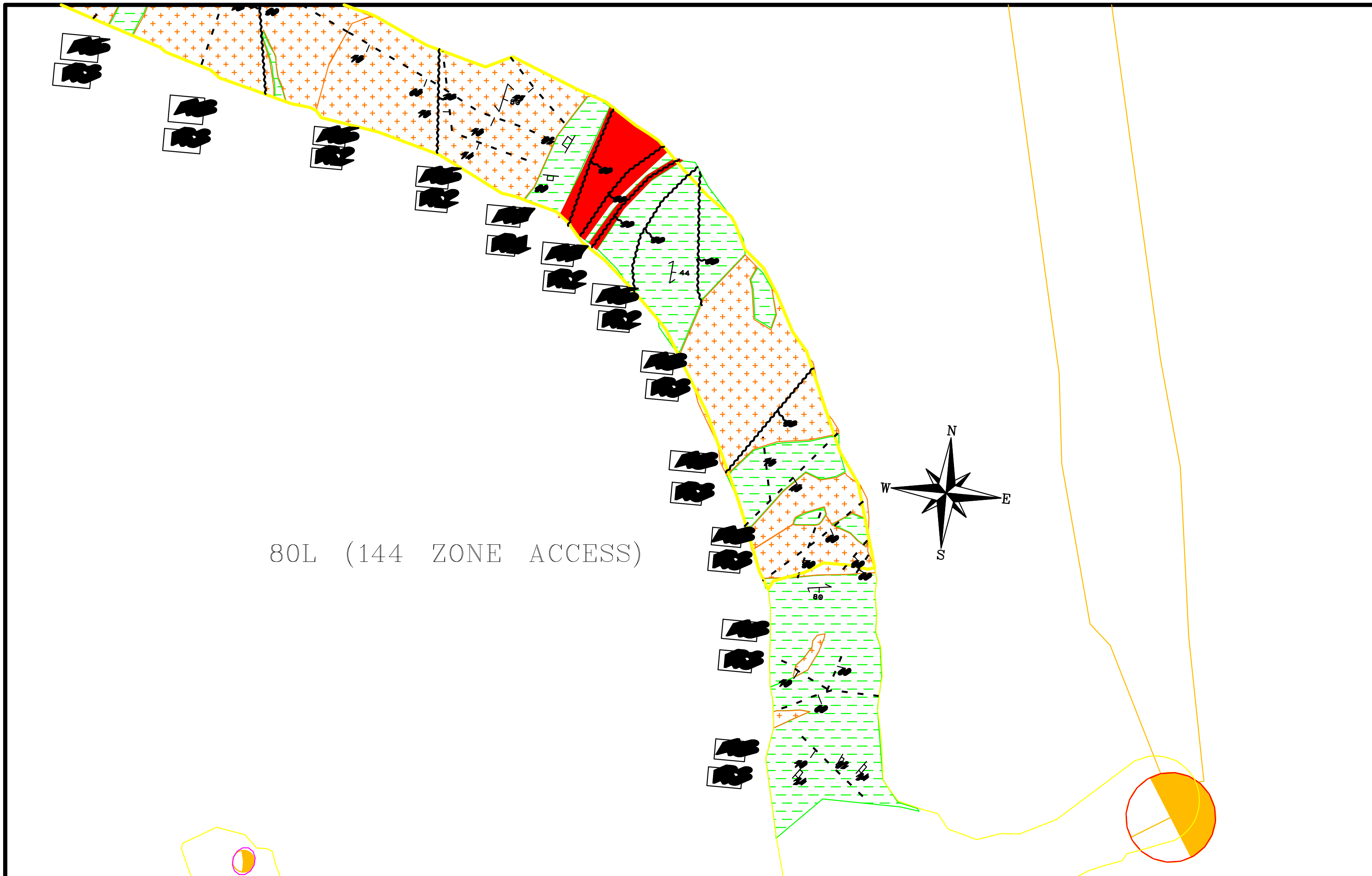
RMR IS 60% NOTE ~20m FROM FACE AS NOT ACCESSABLE. NO FLAT JOINTS. DISCUSSION WITH C.G. THE FACE HAS MORE CLAY AND 50% RMR. THIS AREA IS SUPPORTED WITH MECHANICAL AND MUST RETORQUED AS 50% BELOW STD. AFTER BLAST. THS DRIFT SPAN IS 5.3m(W) X 4.9m(H). NOTE RIGHT WALL.



10

STOP #3: 80L-144 ZONE STH ACCESS DRIFT – HIT WATER "STOPPED"

LOOKING AT ACCESS DRIFT ON 80L TO 144 ZONE STH ORE. THIS DRIFT WAS STOPPED AS PROBE HOLES EXCEEDED 10gpm.



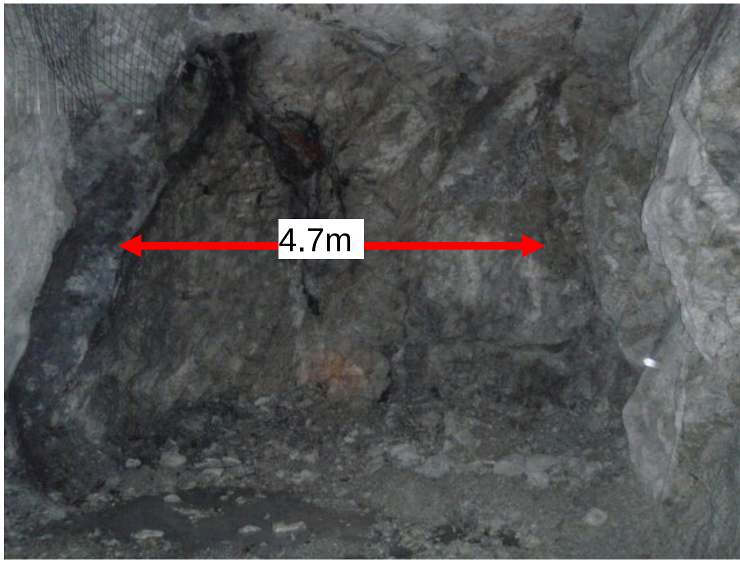
Cameco

RABBIT LAKE OPERATION
EAGLE POINT MINE

TITLE:

EAGLE POINT MINE
80L GEOLOGICAL MAPPING

DATE:	06-Dec-2010	DWG #:
SCALE:	1:250	REVISION:
BY:	KF	FILE:
REVISED BY:		

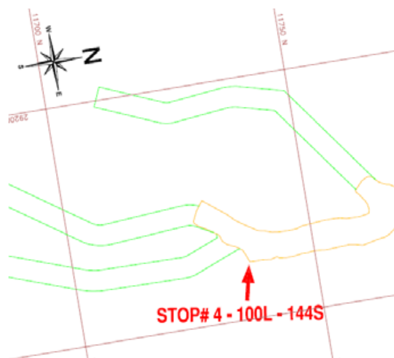


LOOKING AT FACE SPAN IS 4.7m(W) STD SUPPORT + SHOTCRETE WITH RMR OF 40%. FLAT JOINTS IN BACK.

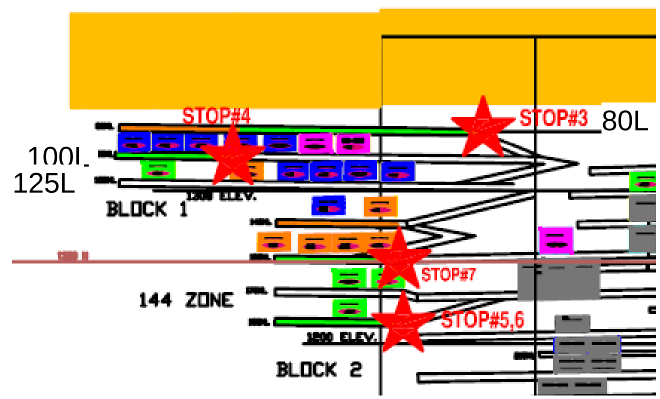
RMR BACK/WALL ALTERED

1) STRENGTH	25-50MPa	4
2) RQD	50%-25%	8
3) SPACING	50-300mm+	15-10
4) CONDITON	SLT-OPN(CLAY)	12-6
5) GRNWTR	DRY-MOIST	10-7
	RATING	49%-35%
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	40%

BACK. SCREEN/STD BOLTING + SHOTCRETE.

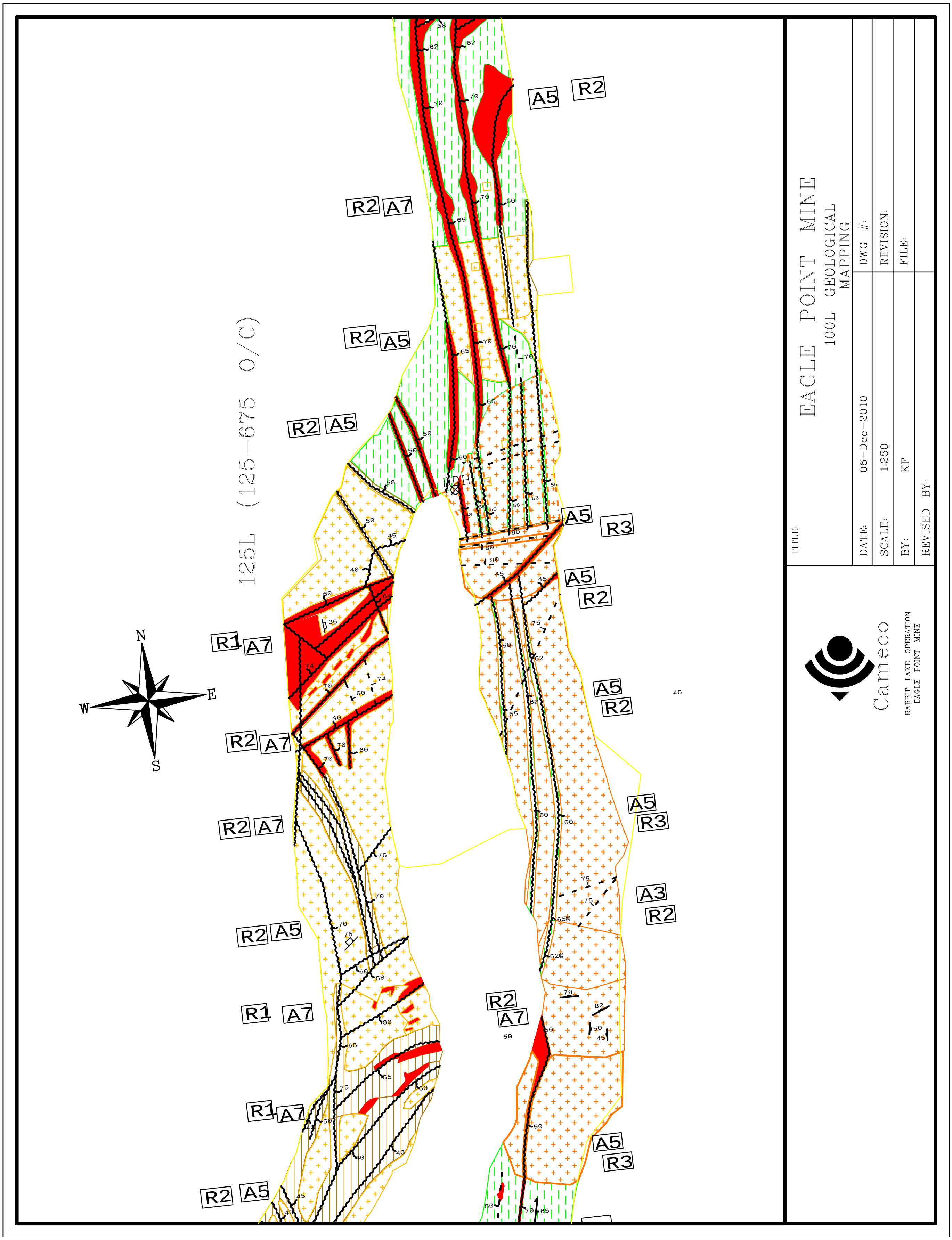


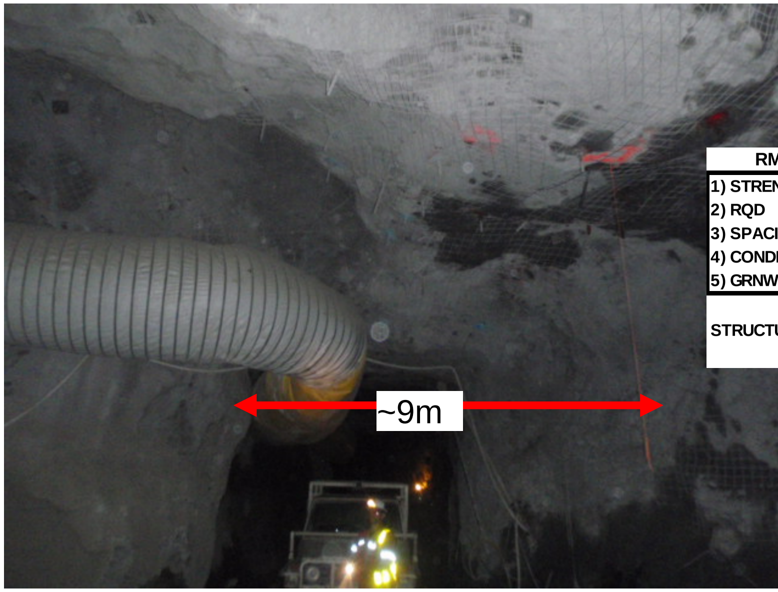
PLAN 100L



STOP #4: 100L-144 ZONE STH O/C(100-125) U/C(80-100)

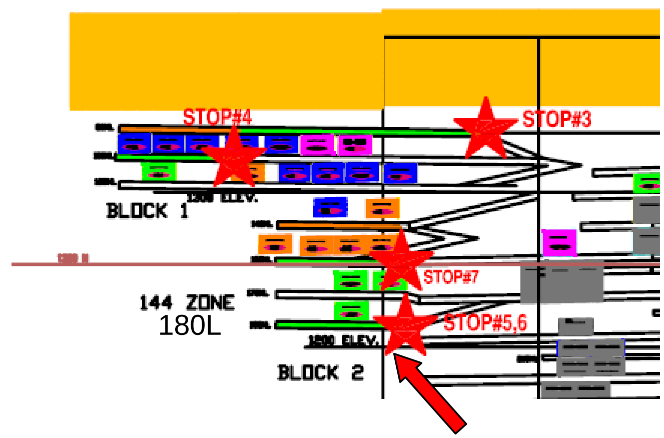
LOOKING AT FACE OF 144 ZONE STH. DRIFT IS 4.7m(W) HAVING A DESIGN RMR=40%. SUPPORT IS COMPRISED OF #6 -2.4m LONG REBAR ON A 1.2m X 1.2m PATTERN WITH 6ft SPLITSETS IN WALL ON 1.2m X 1.2m PATTERN. SCREEN PLACED TO WITHIN 3ft OF FLOOR DUE TO WEAK ROCK MASS. ACCESS WAS SHOTCRETED DUE TO GAMMA.


125L (125-675 O/C)
EAGLE POINT MINE
100L GEOLOGICAL MAPPING
TITLE:
DWG #:
REVISION:
FILE:
DATE: 06-Dec-2010
SCALE: 1:250
BY: KF
REVISED BY:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE



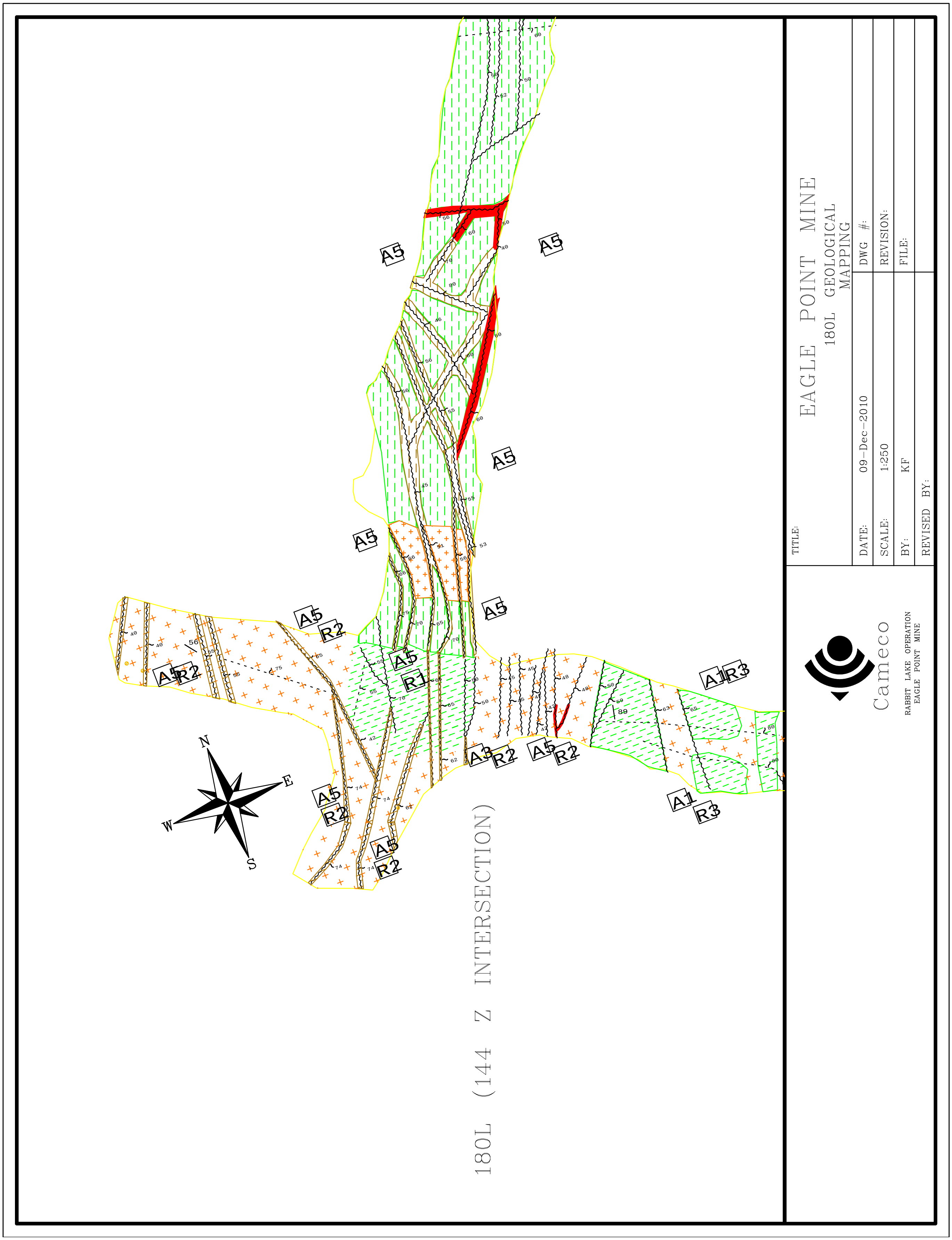
RMR BACK/WALL ALTERED		
1) STRENGTH	50-100MPa	7
2) RQD	50%-25%	8
3) SPACING	50-300mm-	10-5
4) CONDITON	SLT-OPN	12-6
5) GRNWTR	DRY	10
STRUCTURE	RATING	47%-36%
	DESIGN (BACK)	40%

INTERSECTION HAVING SPAN OF ~9m. EMPLOY NINE(9) EXTENSION REBAR (16ft) IN ADDITION TO STD SUPPORT WHICH IS COMPRISED OF 2.4m LONG #6 REBAR ON 1.2m X 1.2m PATTERN WITH SHOTCRETE. ROCK BREAKS WITH MILD HAND PRESSURE.

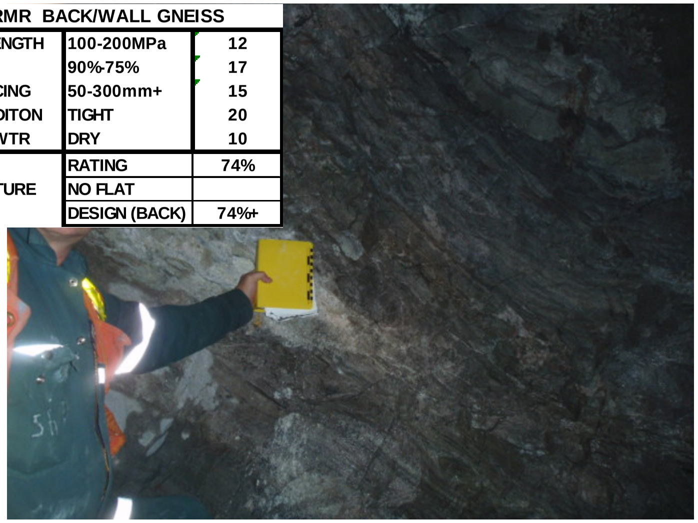


STOP #5: 180L-144 ZONE STH INTERSECTION

LOOKING AT BACK OF INTERSECTION. SPAN IS ~9m WITH 40% RMR. EMPLOYS NINE(9) EXTENSION REBAR (16ft) PLUS STD SUPPORT. THIS AREA WILL BE IN PRODUCTION WITHIN NEXT THREE MONTHS. NOTE CRITICAL SPAN EXCEEDED WITH 40% RMR THEREFORE EXTENSION REBAR + SHOTCRETE IS CRITICAL TO OVERALL STABILITY. AREA WAS ADDRESSED IN MEMO 21/05/2009 BY C. CHERGHEL (APPENDIX). SHOTCRETE IS IMPORTANT TO CONFINE THE OVERALL WEEK MATERIAL AND BOLTS TO ENSURE “DEAD WEIGHT” STABILITY.

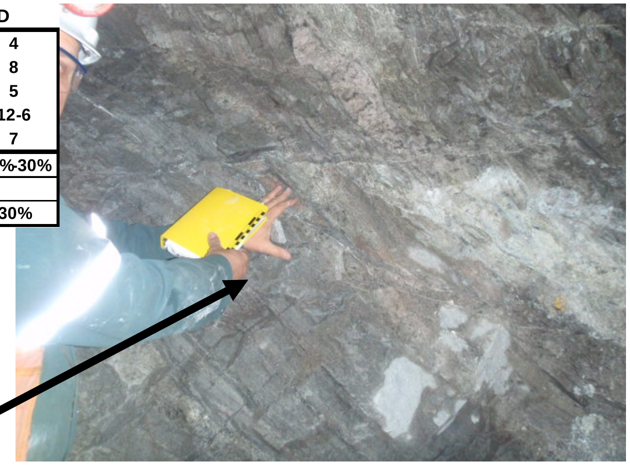

EAGLE POINT MINE
180L GEOLOGICAL MAPPING
TITLE:
DATE: 09-Dec-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
180L (144 Z INTERSECTION)
N
E
S
W
A5
R2
R1
A3
A1
R3

RMR BACK/WALL GNEISS		
1) STRENGTH	100-200MPa	12
2) RQD	90%-75%	17
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT	20
5) GRNWTR	DRY	10
	RATING	74%
STRUCTURE	NO FLAT	
	DESIGN (BACK)	74%+

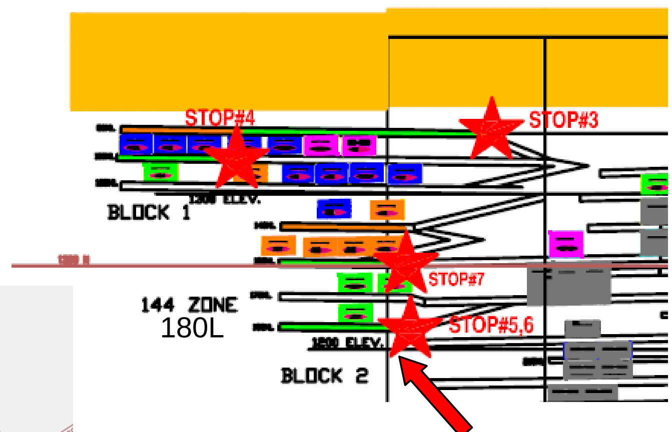


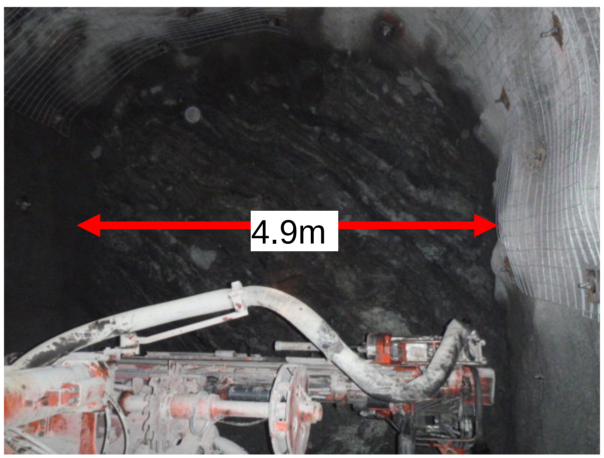
RMR BACK/WALL ALTERED		
1) STRENGTH	25-50MPa	4
2) RQD	50%-25%	8
3) SPACING	<50mm	5
4) CONDITON	SLT-OPN	12-6
5) GRNWTR	MOIST	7
	RATING	34%-30%
STRUCTURE		
	DESIGN (BACK)	30%

LOOKING AT FACE ORE CLAY ALTERED QUARTZITE/PEGMATITE HAVING RIMR OF 30%. NOTE BULK OF FACE(~90%) IS 70%+ RMR.

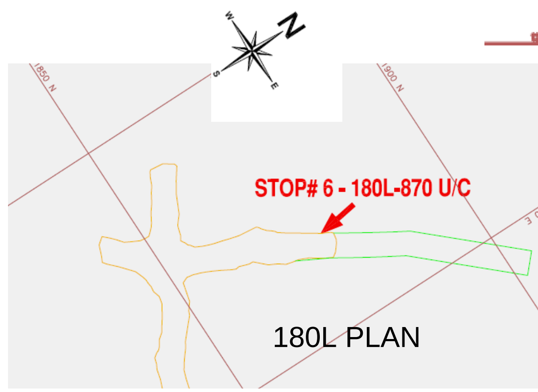


LOOKING AT FACE ORE. QUARTZITE/PEGMATITE HAVING HIGH RMR OF 74%. ~ BULK OF FACE(~90%) IS THIS ROCK MASS REMAINDER ALTERED TO 30% RMR.



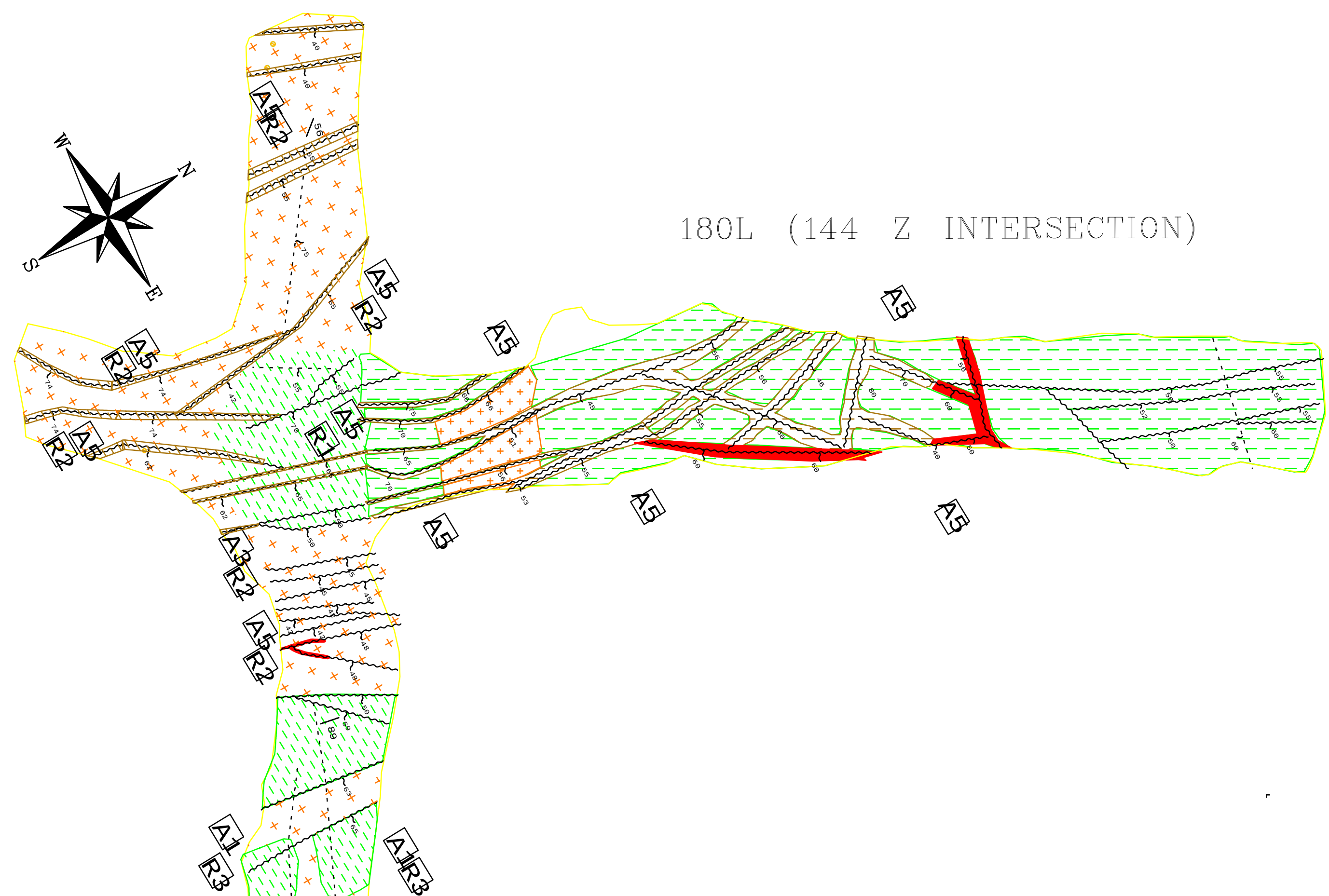


LOOKING AT FACE SPAN IS 4.9m WITH NO FLAT JOINTS FOLIATION DIP 50º/DDR 095º



STOP #6: 180L-144 ZONE STH FACE

LOOKING NORTH AT FACE EMPLOYING STD SUPPORT + SHOTCRETE(50mm). SPAN IS 4.9m. THE BULK OF FACE(~90%) IS GNEISS/PEGMATITE HAVING AN RMR IN EXCESS OF 70% WITH EAST WALL COMPRISED OF ALTERED GNEISS HAVING RMR OF 30%+.

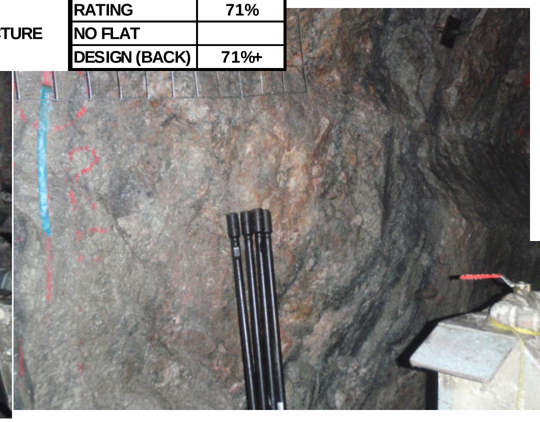


TITLE:	EAGLE POINT MINE 180L GEOLOGICAL MAPPING	
DATE:	04-Jan-2011	DWG #:
SCALE:	1:250	REVISION:
BY:	KF	FILE:
REVISED BY:		

Cameco

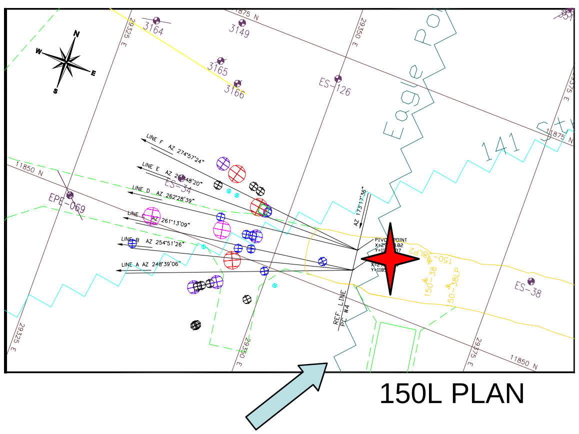
RABBIT LAKE OPERATION
EAGLE POINT MINE

RMR BACK/WALL PEGMATITE		
1) STRENGTH	100-200MPa	12
2) RQD	90%-75%	17
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT	20
5) GRNWTR	MOIST	7
STRUCTURE	RATING	71%
	NO FLAT	
	DESIGN (BACK)	71%+



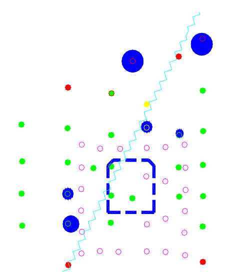
FACE/WALL/BACK IS IN EXCESS OF 70% RMR FOR 5.3m (W) SPAN AND STRUCTURE IS TIGHT. NO VISIBLE CONCERN FOR WATER, HOWEVER, GEOLOGY SHOWS EAGLE POINT FAULT ~ 4m INTO FACE AS WELL AS PROBE DRILLING WHICH IS STD PRACTICE ENCOUNTERED 20GPM OF WATER.

AREA GROUTED AS WATER IN EXCESS OF 20GPM ENCOUNTERED WITH PROBE DRILLING ~4m BEYOND EXISTING FACE. PROCEDURES ARE TO PROBE DRILL EACH ROUND AND IF IN EXCESS OF 5GPM TO STOP AND GROUT. SPAN IS 5.3m (W) X 4.9m (H). PRESENTLY IS MUCH LESS THAN 5gpm/DRY AND ROUND IS ABLE TO BE TAKEN.

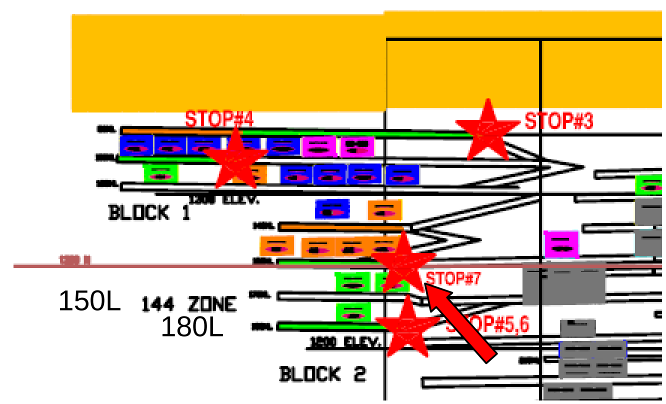


150L PLAN

EAGLE POINT FAULT * CONCERN WATER/POOR GROUND



GROUT COVER

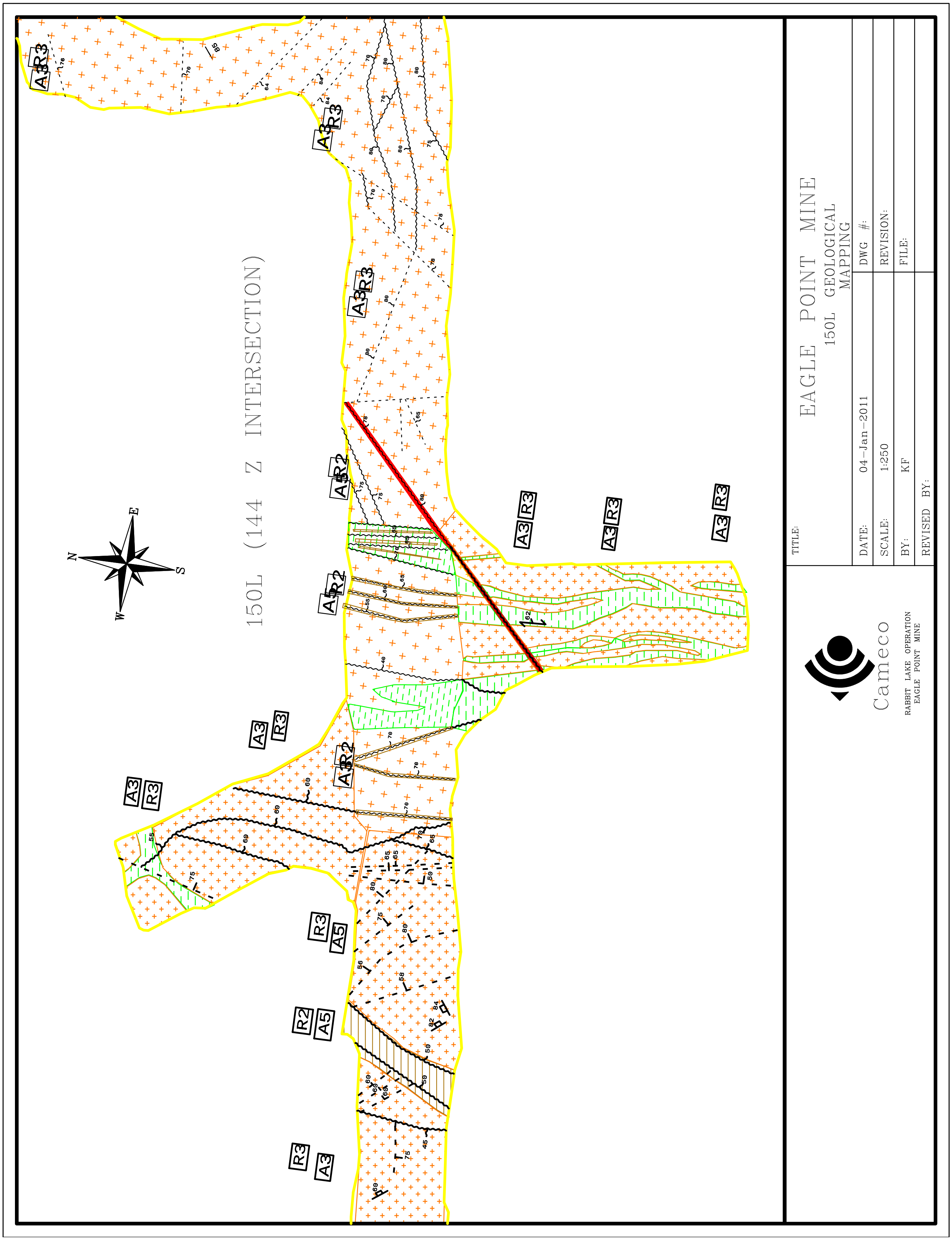


150L
180L

STOP #7: 150L – 144 ZONE RAMP TO ORE.

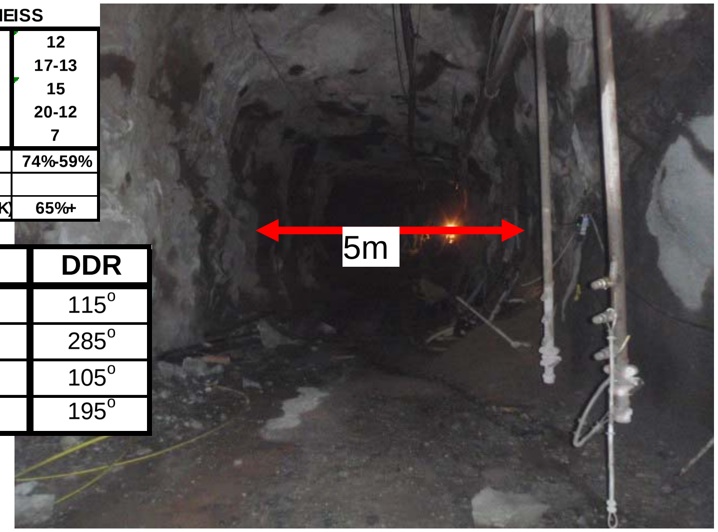
16

GROUT COVER ON 150L – 144 ZONE. NOTE RMR IS IN EXCESS OF 70% RMR HIT WATER 5m AHEAD POOR GROUND. PROXIMITY OF EAGLE POINT FAULT.


150L (144 Z INTERSECTION)
N
E
S
W
A3 R3
A3 R2
R3 A5
R2 A5
R3 A3
TITLE:
EAGLE POINT MINE
150L GEOLOGICAL MAPPING
DWG #:
REVISION:
FILE:
DATE: 04-Jan-2011
SCALE: 1:250
BY: KF
REVISED BY:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE

RMR BACK/WALL GNEISS		
1) STRENGTH	100-200MPa	12
2) RQD	75%-50%+	17-13
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT-SLT	20-12
5) GRNWTR	MOIST	7
STRUCTURE	RATING	74%-59%
	NO FLAT	
	DESIGN (WL/BK)	65%+

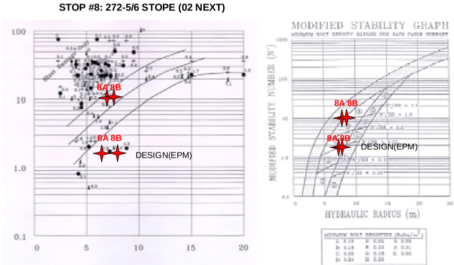
JS	DIP	DDR
1	40°	115°
2	50°	285°
3	65°	105°
4	80°	195°



OVERCUT ON 252L LOOKING STH.

LONGHOLE STOPE FROM 252-272L LOOKING AT CABLED HW. STRIKE LENGTH EXISTING IS 28m WITH HW (H) CENTER TO CENTER OF 30m. HR=7.2m. RMR=65%+ OR EQUIVALENT Q'=10.3 OR N=A X B X C. SLOPE INCLINED AT 60o THEREFORE C=8-6*COS60=5, B=0.2 CROSS CUTTING,A=1.N=10.3 X 1 X 0.2 X 5 = 10.3. PROPOSED TO EXTEND HW TO STRIKE OF 32m RESULTING IN HR=7.7m. NOT MAJOR INCREASE IN SLOUGH EXPECTED AS THE HW RMR(65%) IS BETTER THAN INITIALLY DESIGNED(48%)

STOP #8: 272-5/6 STOPE (02 NEXT)

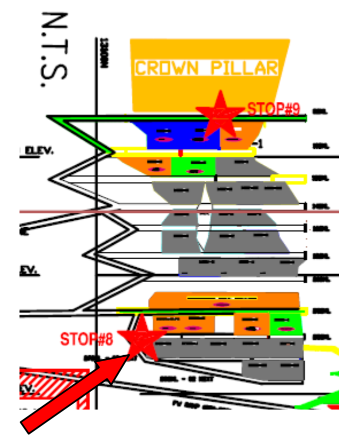


HW OBSERVATIONS - EAGLE POINT MINE - JUNE 1-4/09

STOP	LOCATION	Q'	RMR	A	B	C	N	DIMENSIONS HW (m x m)	HYD. RD. (m)	ELOS (m)
8A	272-5/6	10.3	65%	1	0.2	5	10.3	28L X 30H	7.2m	NOT CMS
8B	272-5/6	10.3	65%	1	0.2	5	10.3	32L X 30H	7.7m	PROPOSED

DESIGN WAS RMR=48% Q=1.6, N=1.6 (EPM- CORE)

252L PLAN



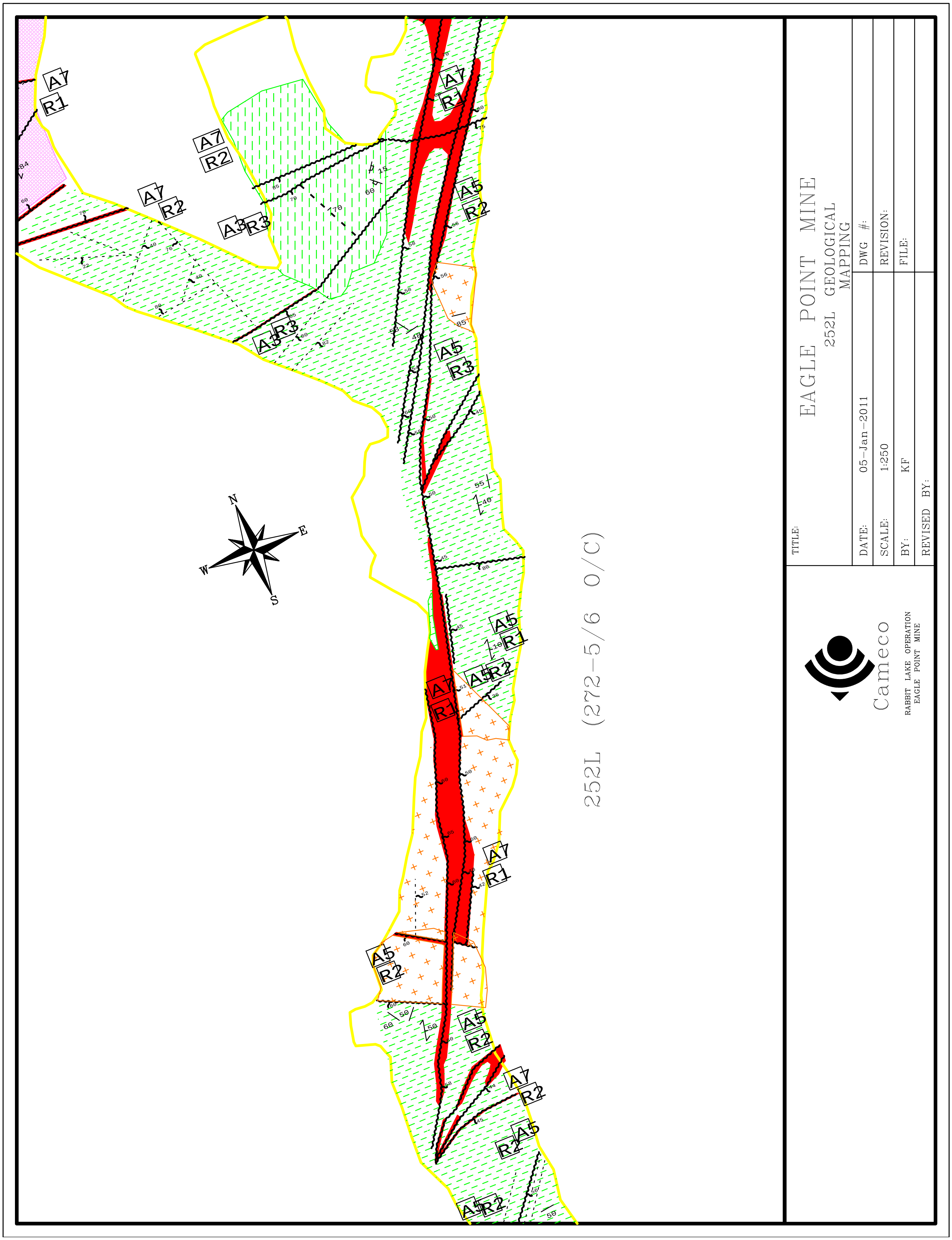
31

HW OBSERVATIONS – EAGLE POINT MINE – JUNE 1-4/09

STOP #8: 252L O/C, 272-5/6 STOPE (O2 NEXT)

LOOKING NORTH AT 272-5/6 LONGHOLE STOPE ON 252L. THE RMR IS BETTER(65%+) THAN DESIGN(48%) AND IS REFLECTED IN THE OVERALL HW STABILITY. AN OPTION IS TO EXTEND THE STRIKE LENGTH FROM EXISTING 28m TO 32m AND MINIMAL/NO INCREASE IN SLOUGH IS EXPECTED. NOTE THAT THE STOPE WAS MAPPED/CORE LOGGED AS A5/R2 ~ Q'=1.66 `RMR=48% NOTE NOT DIRECT EQUIVALENT SHOULD ASSESS IN FIELD IS CONSERVATIVE.

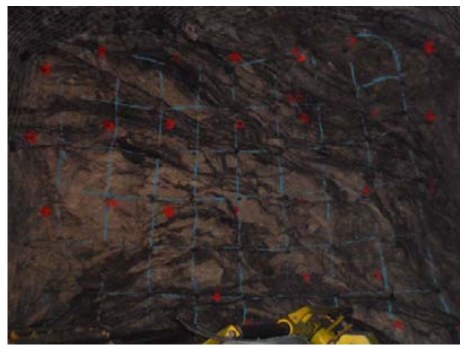
17


EAGLE POINT MINE
252L GEOLOGICAL MAPPING
TITLE:
DWG #:
REVISION:
FILE:
DATE: 05-Jan-2011
SCALE: 1:250
BY: KF
REVISED BY:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
252L (272-5/6 O/C)
N
E
S
W

LOOKING WEST AT FACE.

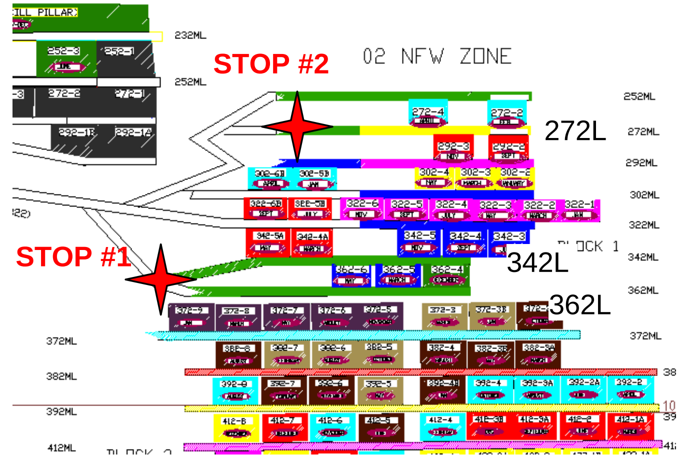
RMR BACK/WALL GNEISS

1) STRENGTH	50-100MPa	7
2) RQD	90%-75%-	17-13
3) SPACING	50-300mm+	15
4) CONDITON	SLT	12
5) GRNWTR	MOIST	7
	RATING(WALL)	58%-54%
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	45%+

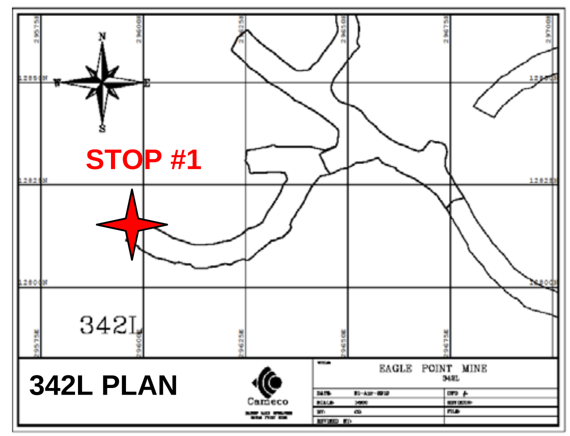


FLAT JOINTS IN FACE/BACK

BACK SHOWING BAGGED LOOSE. #9 GAUGE ABLE TO SUPPORT 1.9 tonne WHICH IS EQUIVALENT TO 0.5m+ OF BAGGED EMPLOYING 1.2m X 1.2m PATTERN WITH 2.5 SG ROCK. NOTE INTACT SG EMPLOYED SHOULD USE BULKED.



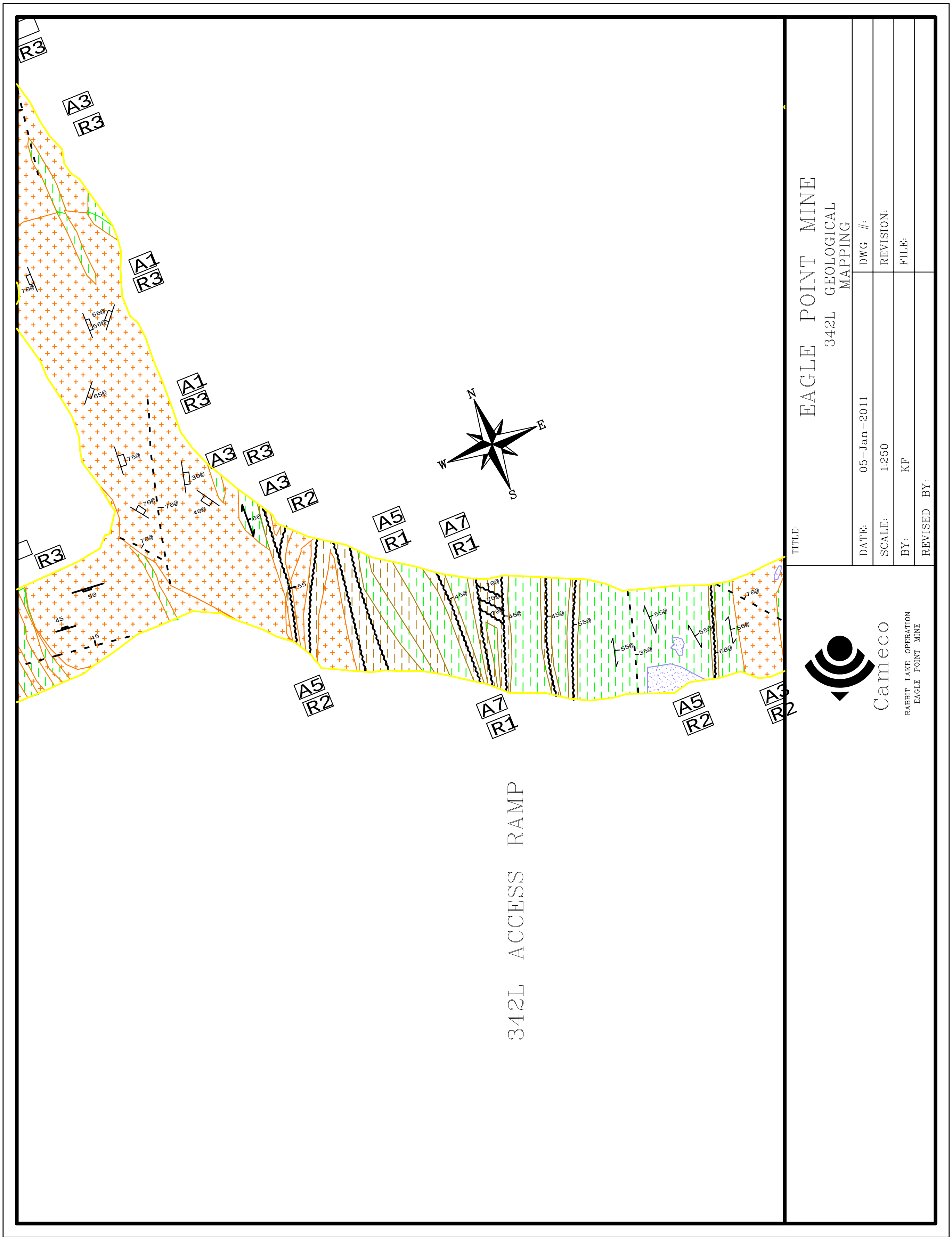
02 NEXT FW ZONE



STOP #1: 342L RAMP – (02 NEXT FW)

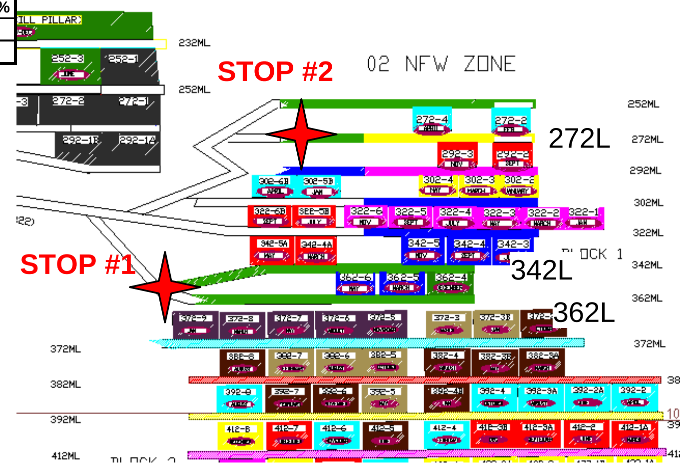
LOOKING WEST AT FACE. RMR DESIGN IS 55%-10% FOR FLAT JOINTS (45%) WITH BACK SPAN OF 6.5m. BACK SUPPORT COMPRISED OF 2.4m LONG #6 REBAR ON 1.2m X 1.2m PATTERN WITH GALVANIZED WWM #9 GAUGE SCREEN WITH SIMILAR PATTERN ON WALLS OTHER THAN BOTTOM ROW OF SPLITSETS 1.5m FROM FLOOR. NOTE NO STRESS FAILURE OBSERVED/WEAK ROCK MASS FAILURE/SLAB.

6


EAGLE POINT MINE
342L GEOLOGICAL MAPPING
TITLE:
DATE: 05-Jan-2011
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
342L ACCESS RAMP
N
E
S
W

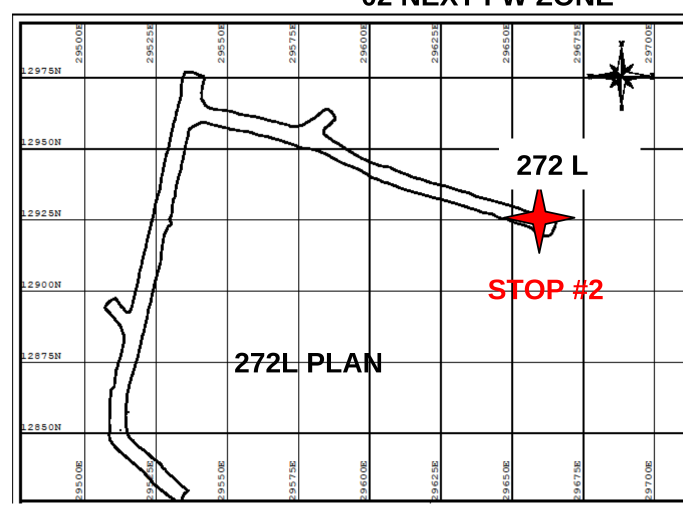
RMR BACK/WALL GNEISS

1) STRENGTH	100-200MPa	12
2) RQD	90%-75%	17
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT-SLT	20-12
5) GRNWTR	DRY	10
	RATING(WALL)	74%-66%
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	60%+



LOOKING EAST AT FACE.

02 NEXT FW ZONE



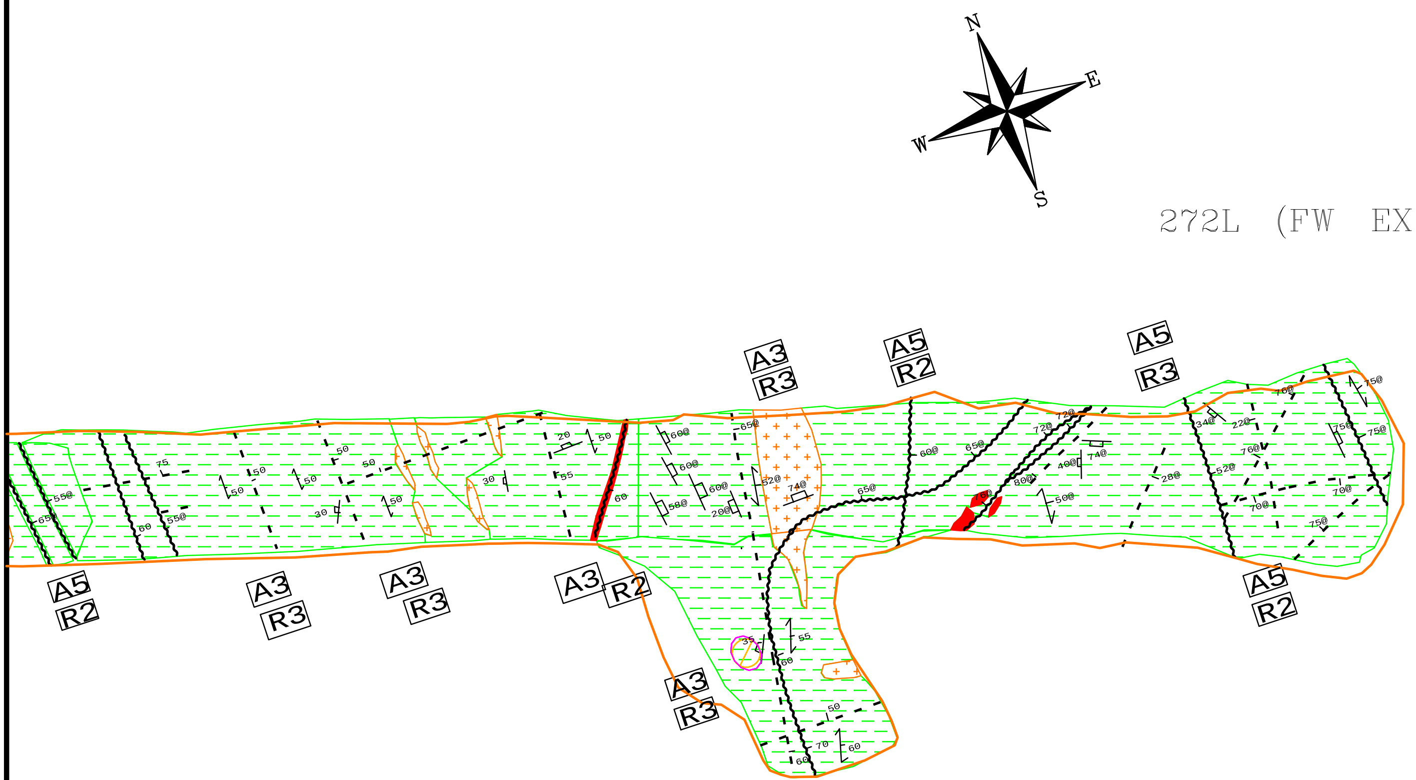
FLAT JOINTS IN FACE/BACK

STOP #2: 272L DEVELOPMENT (WASTE) "02 NEXT FW ZONE")

8

LOOKING EAST AT FACE. RMR DESIGN IS 70%-10% FOR FLAT JOINTS (60%) WITH BACK SPAN OF 7m. THIS IS DEEPEST LATERAL DEVELOPMENT AT THIS POINT AND IS LOCATED IN THE HW OF THE "02 NEXT FW ZONE". NO GRAPHITE. THIS AREA IS WIDER THAN NORMAL DUE TO RAISE IN AREA FROM 272-322L.

272L (FW EXHAUST DRIFT)



RABBIT LAKE OPERATION
EAGLE POINT MINE

TITLE: EAGLE POINT MINE 272L GEOLOGICAL MAPPING			
DATE:	07-Dec-2010	DWG #:	
SCALE:	1:250	REVISION:	
BY:	KF	FILE:	
REVISED BY:			

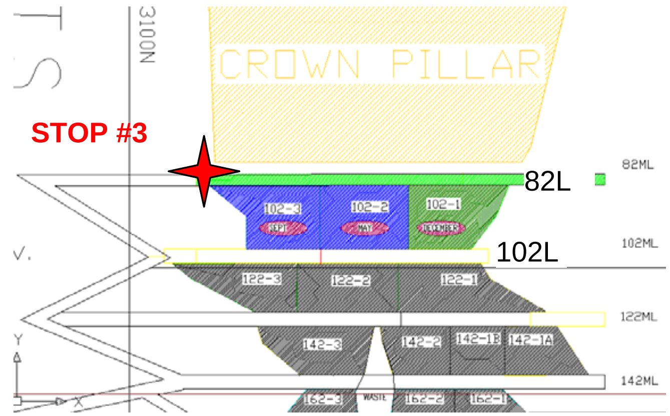
FLAT JOINTS IN FACE/BACK

RMR BACK/WALL GNEISS

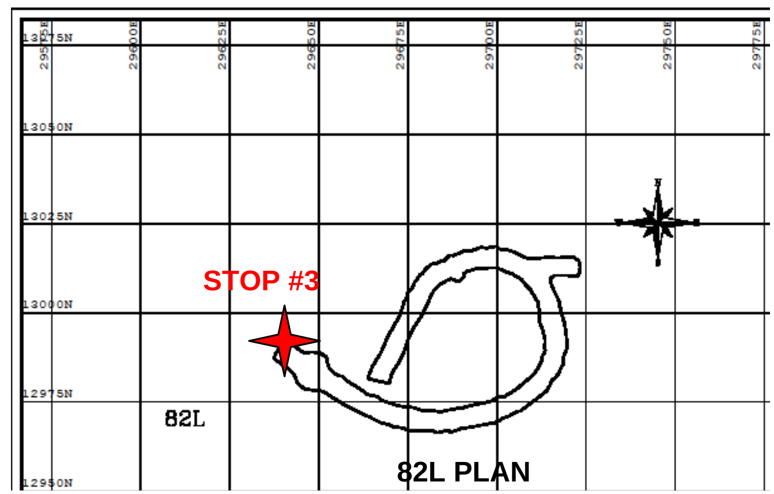
1) STRENGTH	100-200MPa	12
2) RQD	90%-75%-	17
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT-SLT	20-12
5) GRNWTR	DRY	10
	RATING(WALL)	74%-66%
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	60%+

JS	DIP	DDR
1	45^o	105^o
2	80^o	220^o
3	75^o	250^o

TREND = 090^o



02 NEXT EXTENSION ZONE



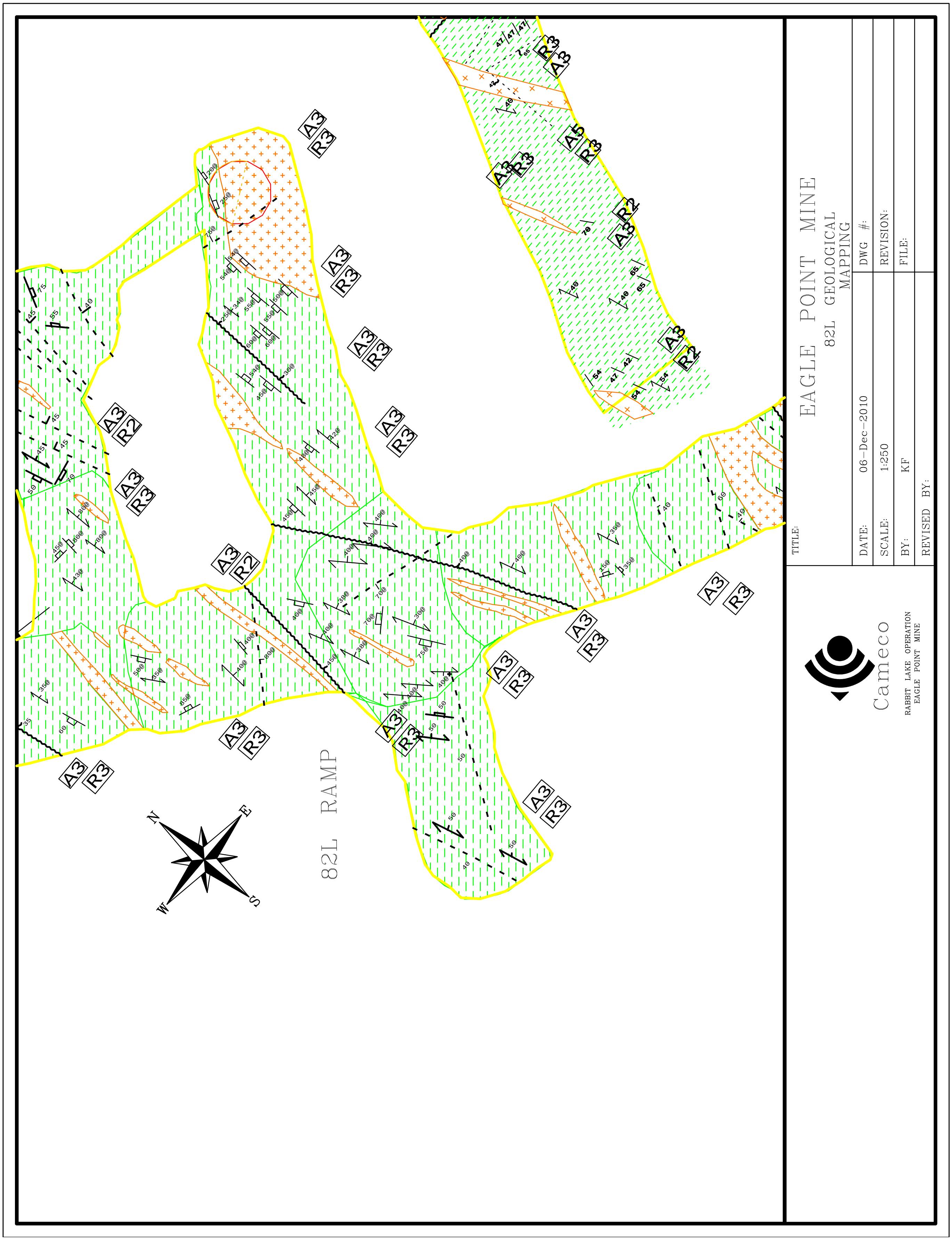
82L PLAN

LOOKING NORTHWEST
AT FACE.

STOP #3: 82L, O2 NEXT O/C (82-102L)

LOOKING NORTHWEST AT FACE. CROWN PILLAR ABOVE (~50m+). OVERCUT DEVELOPMENT WITH STOPES BELOW FROM 82L-102L. ROCK MASS IS 60% IN WALL AND 50% IN BACK DUE TO FLAT JOINTS. SPAN IS 6m. SPLITSETS IN WALL WITH BACK REBAR. JOINTING IN BACK RESULTS IN SLIDING FAILURE “MINIMAL CONCERN”. AREA IS DRY. MINED UNDER GROUT COVER. NOTE CROWN ABOVE (LAKE).

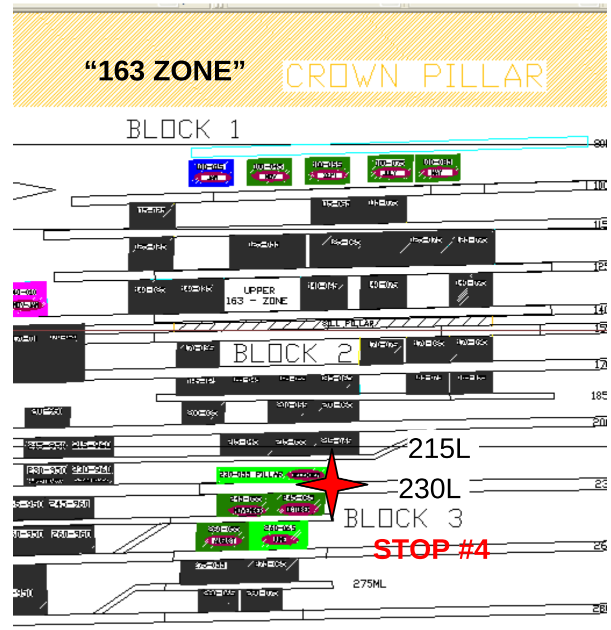
10


EAGLE POINT MINE
82L GEOLOGICAL MAPPING
TITLE:
DATE: 06-Dec-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
82L RAMP

LOOKING NE AT FACE.

RMR BACK/WALL GNEISS

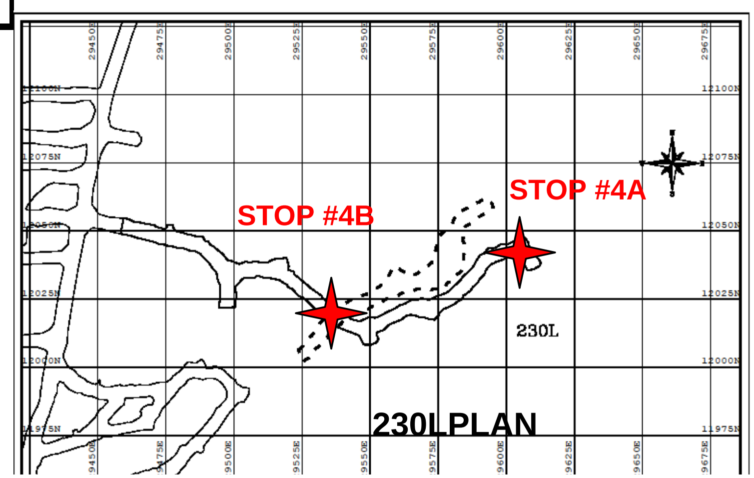
1) STRENGTH	100-200MPa	12
2) RQD	90%-75%-	17
3) SPACING	50-300mm+	15
4) CONDITON	SLT	12
5) GRNWTR	DRY	10
	RATING(WALL)	66%
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	56%+



JS	DIP	DDR
1	50^o	060^o
2	52^o	028^o
3	25^o	287^o

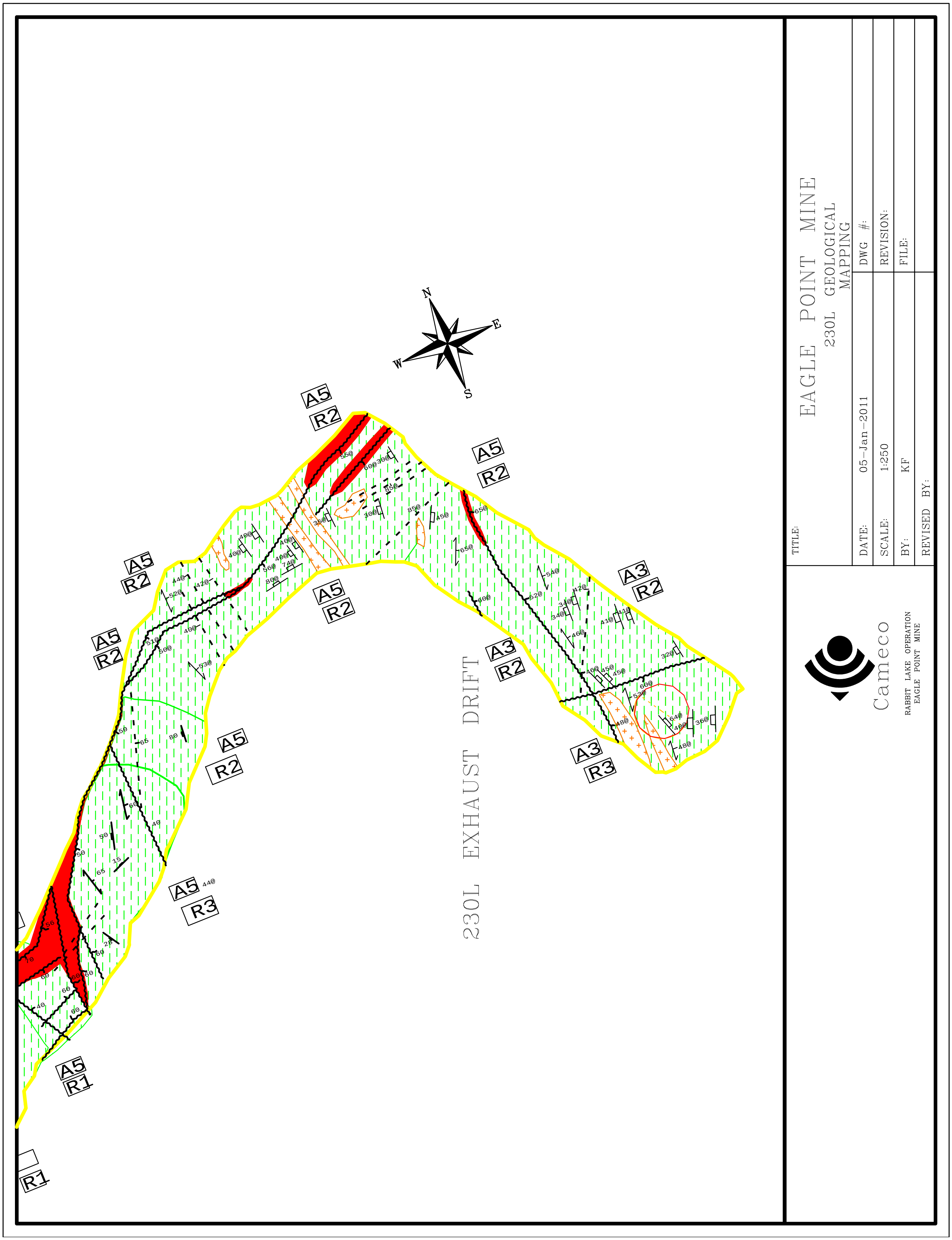
TREND = 050^o

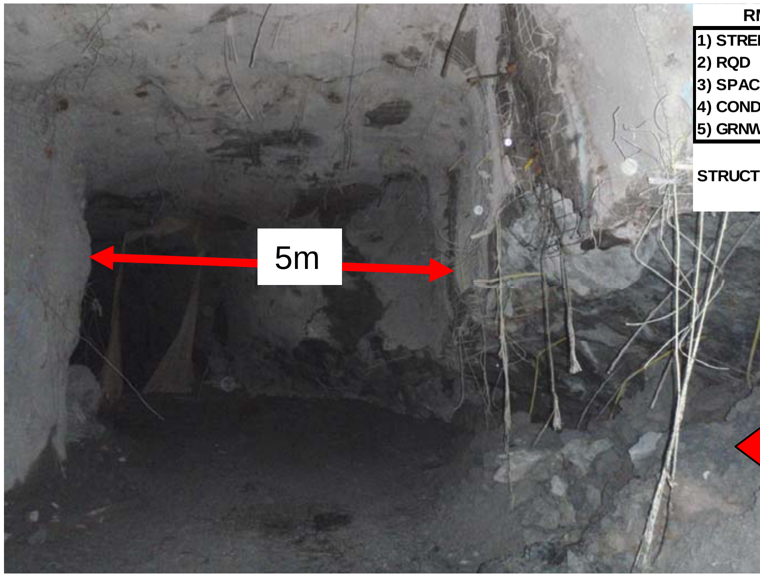
STH WALL (HW) PARALLEL JOINTS TO PROPOSED STOPE (230-245L). DIP AT 50º.



STOP #4B: 230L, O/C 163 ZONE (230L-245L)

LOOKING NORTH EAST AT FACE. BACK SPAN IS 5.5m WITH RMR OF 56% IN BACK DUE TO FLAT JOINTS. WALL IS 66% WITH JOINTS PARALLEL TO HW. SUPPORT IS COMPRISED OF #6 REBAR IN BACK AND WALLS AS NOT WASTE DEVELOPMENT (PROXIMITY OF ORE).. SG=2.3. BOTTOM ROW IS SPLITS WITH SCREEN ~ 1.5m OFF FLOOR. BLOCKY GROUND, HOWEVER, SLIDING WEDGES ONLY.

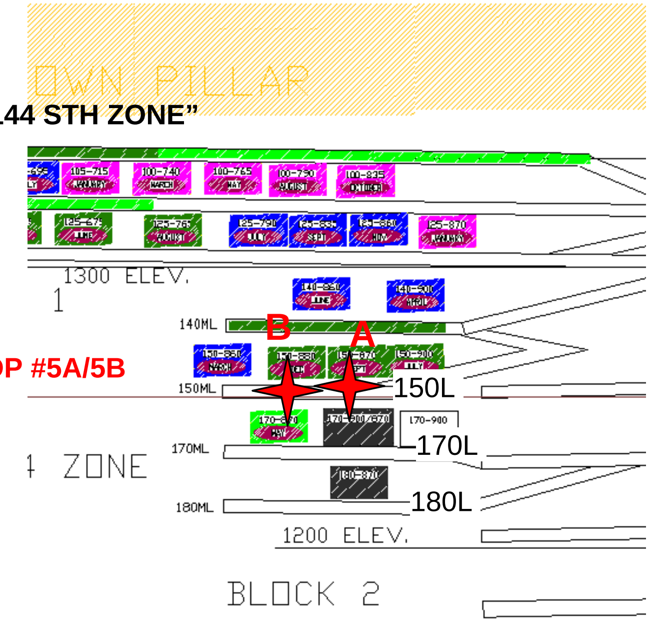

TITLE:
EAGLE POINT MINE
230L GEOLOGICAL MAPPING
DATE: 05-Jan-2011
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
230L EXHAUST DRIFT
N
E
S
W
A5
R2
A3
R3
R1



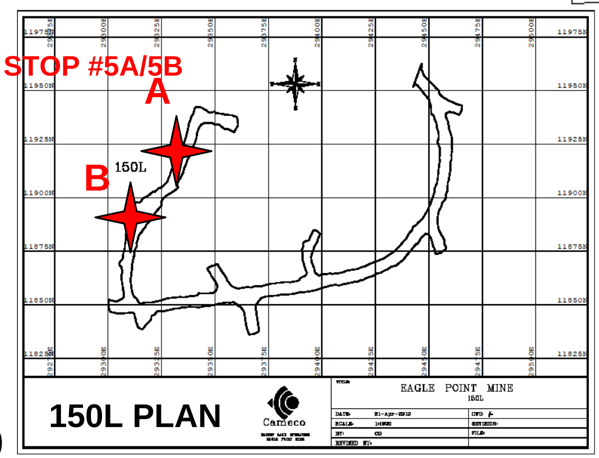
TIGHT FILL
REQ'D

LOOKING NE WITH HW (RIGHT WALL) THAT PROPOGATED FROM 170L BELOW. CAVE RESTRICTED BY SUPPORT ON LEVEL.

RMR	BACK/WALL	ALT PEG
1) STRENGTH	25-50MPa	4
2) RQD	90%-75%	17
3) SPACING	50-300mm+	15
4) CONDITON	SLT-OPN(CLAY)	12-6
5) GRNWTR	DRY	10
	RATING(WALL)	58%-52%
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	45%



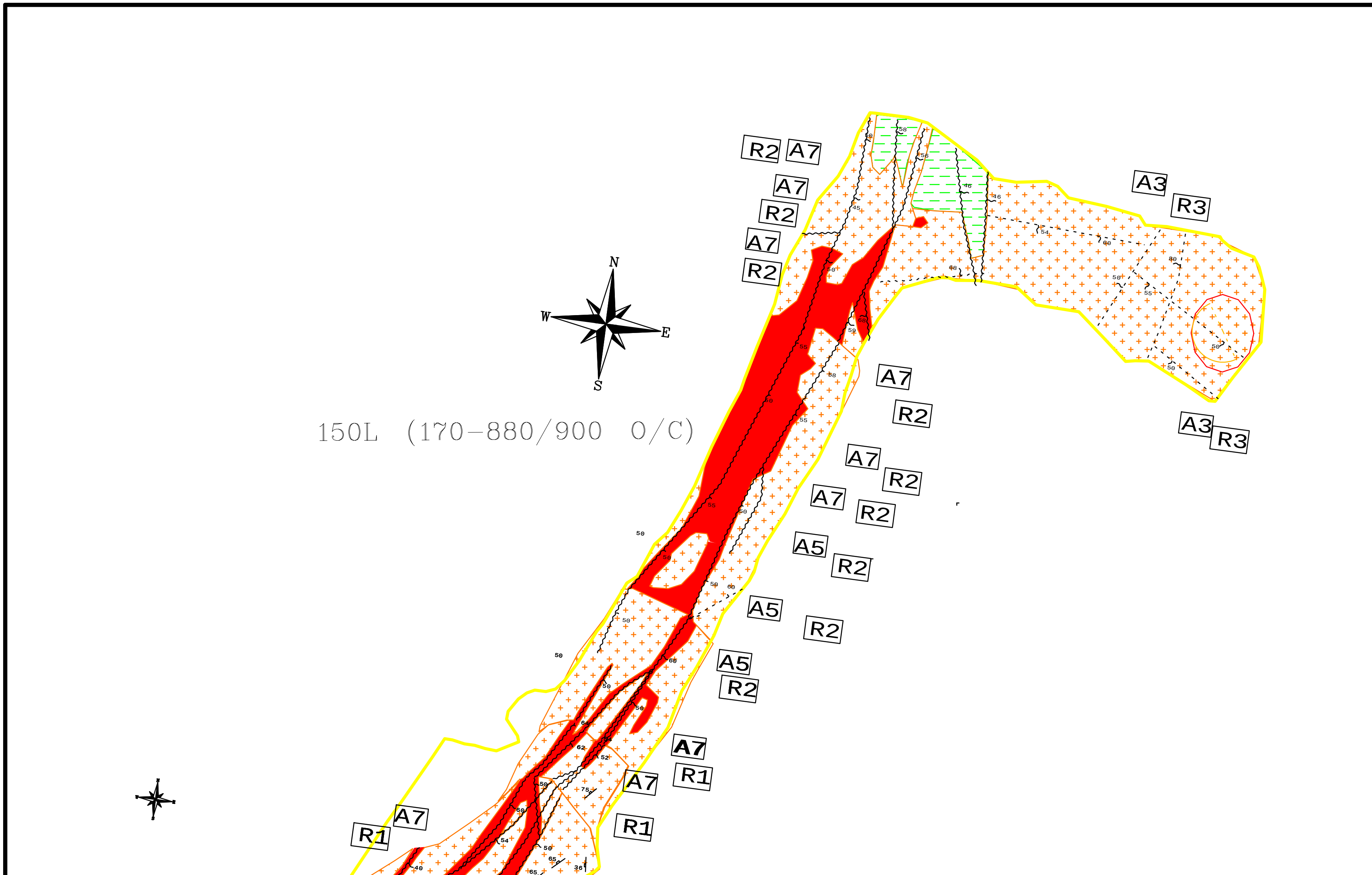
HW STRUCTURE CONTROLLING HW SLOUGH



STOP #5A: 150L, O/C 144 STH – 170 880/900 STOPE (150-170)

LOOKING NORTH EAST. SPAN IS 5m WITH RMR OF 45%. HW (RIGHT WALL) HAS FAILED ~3m+ ON LEVEL BUT MIGRATION ABOVE LEVEL RESTRICTED BY THE CABLES IN PLACE. SUPPORT IS COMPRISED OF STD SUPPORT + 2-3" OF SHOTCRETE + 12m CABLES ON A 2m(BETWEEN RINGS) X 2.5m (TOE SPACING) PATTERN. PLATES NOT COLLARED IN BACK ON CABLES, HOWEVER, NO MOVEMENT OF BACK OBSERVED. CONTROL OF HW SLOUGH IS LARGELY PARALLEL TO A THROUGH GOING SHALLOW DIPPING STRUCTURE . HW IS TIGHT FILLED AND AREAS WHERE PRESENTLY INACCESSABLE WILL BE TIGHT FILLED.

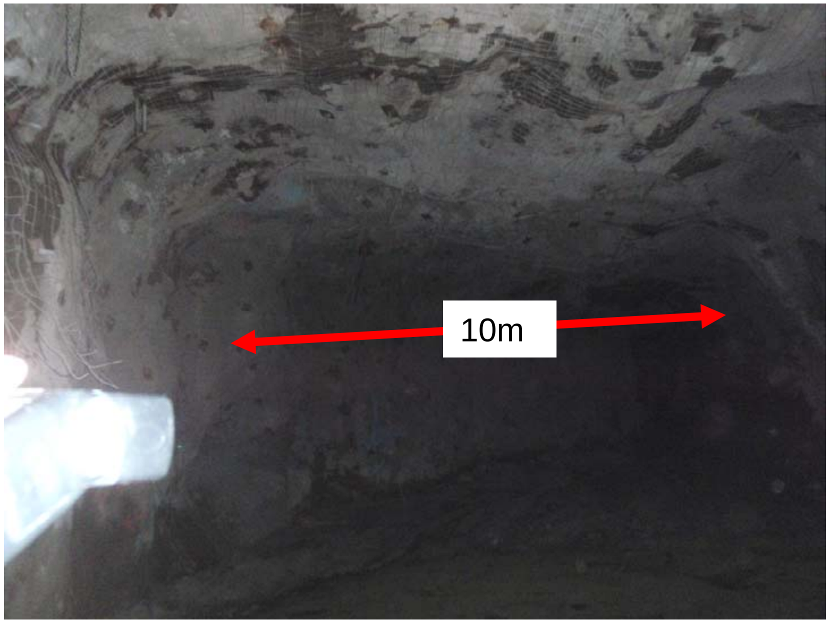
21

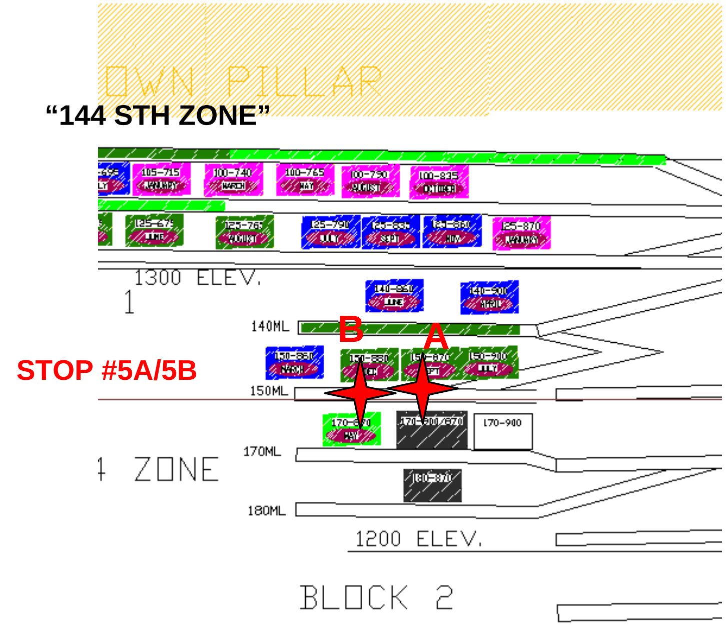


Cameco

RABBIT LAKE OPERATION
EAGLE POINT MINE

TITLE: EAGLE POINT MINE 150L GEOLOGICAL MAPPING	
DATE: 04-Jan-2011	DWG #:
SCALE: 1:250	REVISION:
BY: KF	FILE:
REVISED BY:	





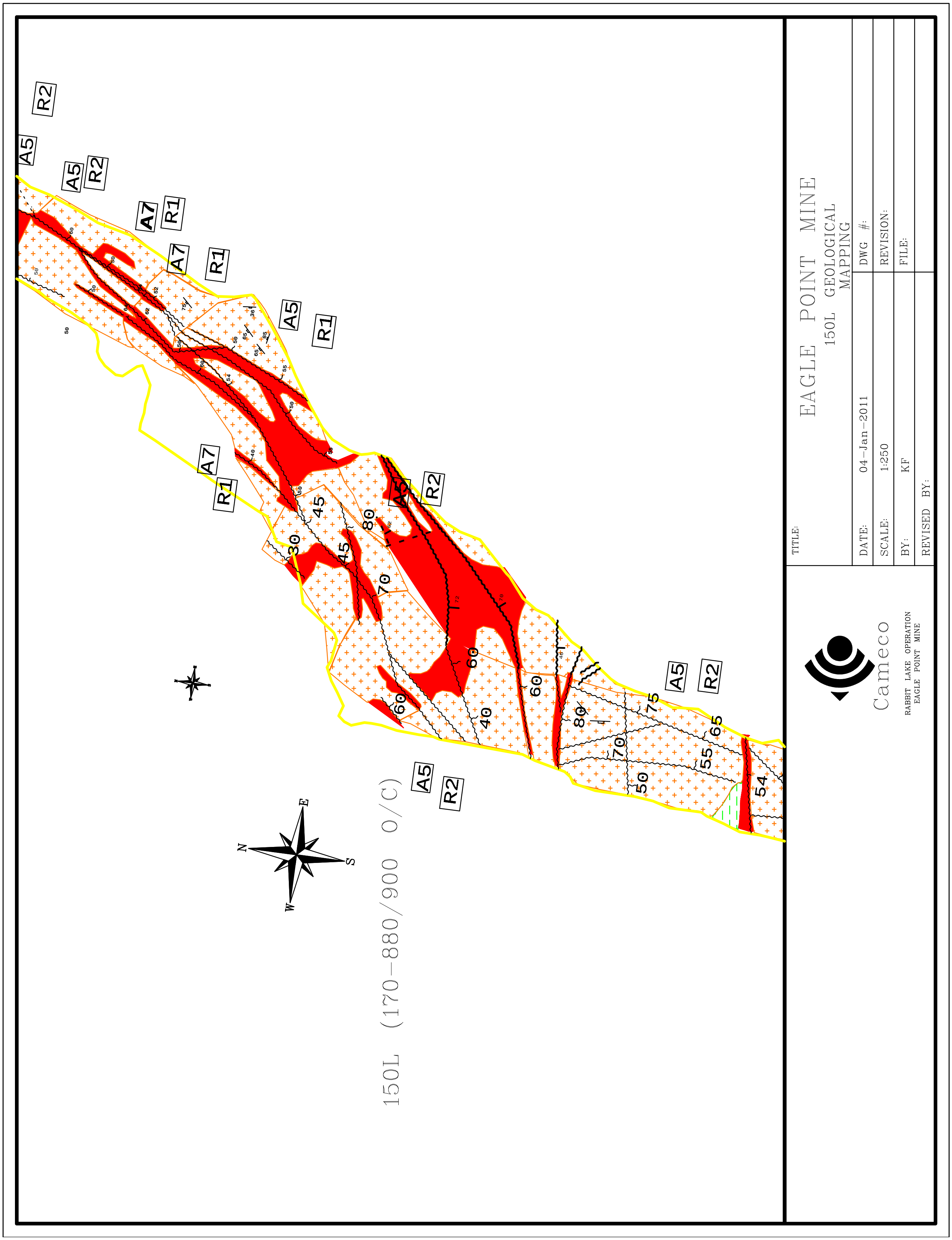
LOOKING NE WITH HW (RIGHT WALL) THIS STOPE BLOCK IS NEXT TO BE MINED FROM 150-170L. BACK IS STD SUPPORT + CABLES+ SHOTCRETE. STABLE WITH SUPPORT. CONCERN IS 10m SPAN AND IF FURTHER HW FAILURE MAY OVERIDE CABLE SUPPORT.

RMR BACK/WALL ALT PEG

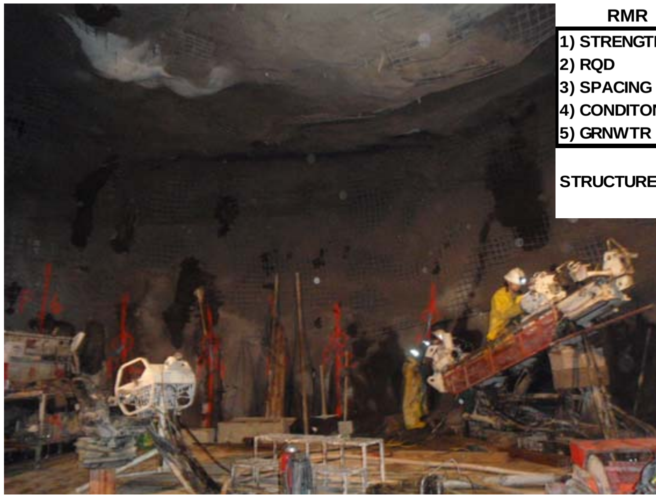
1) STRENGTH	25-50MPa	4
2) RQD	90%-75%	17
3) SPACING	50-300mm+	15
4) CONDITON	SLT-OPN(CLAY)	12-6
5) GRNWTR	DRY	10
	RATING(WALL)	58%-52%
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	45%

STOP #5B: 150L, O/C 144 STH – 170 870 STOPE (150-170)

LOOKING NORTH EAST. SPAN IS 10m WITH RMR OF 45%. THE AREA IS POTENTIALLY UNSTABLE WITH LOCAL SUPPORT BUT STABLE WITH CABLES IN PLACE + STD SUPPORT. NOTE THAT SPAN OF 10m COUPLED WITH ANY FURTHER INCREASE IN HW SLOUGH WILL RESULT IN BACK SPANS >>10m AND MAY OVER RIDE THE CABLE SUPPORT IN PLACE. **THIS SHOULD BE ASSESSED.**

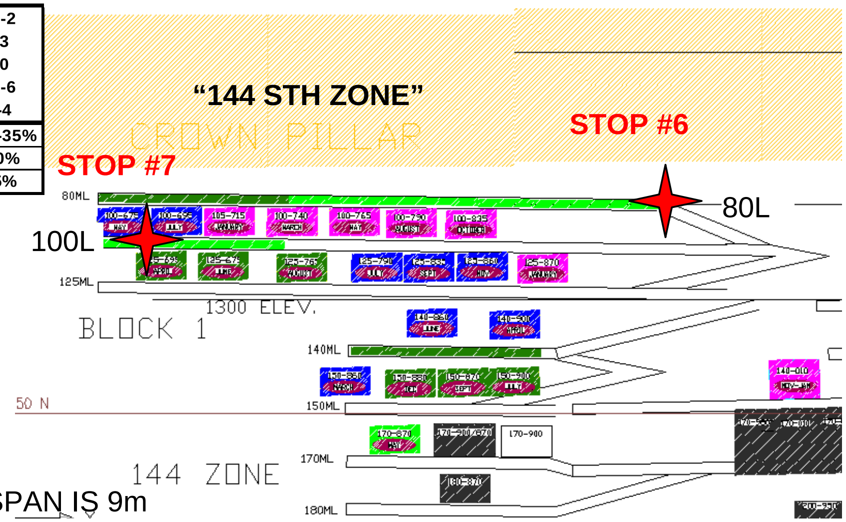


EAGLE POINT MINE
150L GEOLOGICAL MAPPING
TITLE:
DWG #:
REVISION:
FILE:
DATE: 04-Jan-2011
SCALE: 1:250
BY: KF
REVISED BY:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
150L (170-880/900 O/C)
A5
R2
A7
R1
N
E
S
W

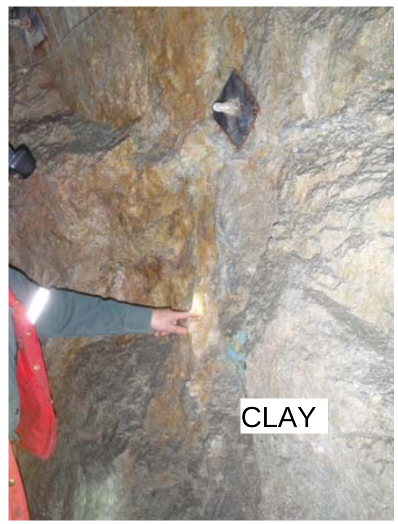


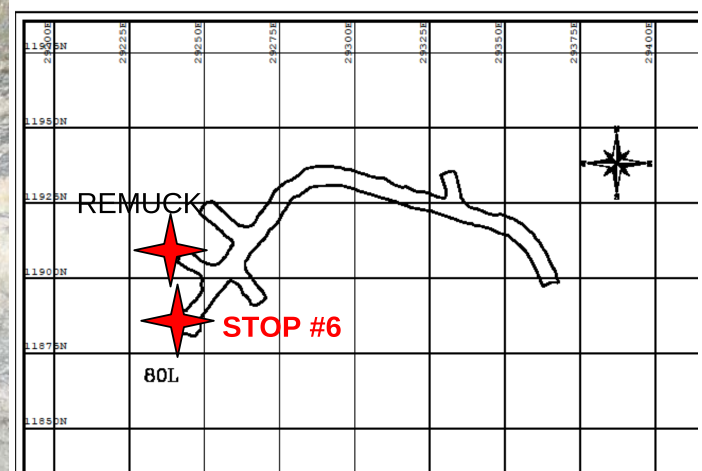
RMR BACK/WALL ALT PEG

1) STRENGTH	100-10MPa	12-2
2) RQD	75%-50%	13
3) SPACING	50-300mm	10
4) CONDITON	TIGHT-OPN	20-6
5) GRNWTR	MOIST-FLOW	7-4
STRUCTURE	RATING(WALL)	62%-35%
	FLAT	-10%
	DESIGN (BACK)	45%



LOOKING WEST AT THE 80L, WASTE DEVELOPMENT BAY. SPAN IS 9m
ESTIMATED RMR IS 45%+.



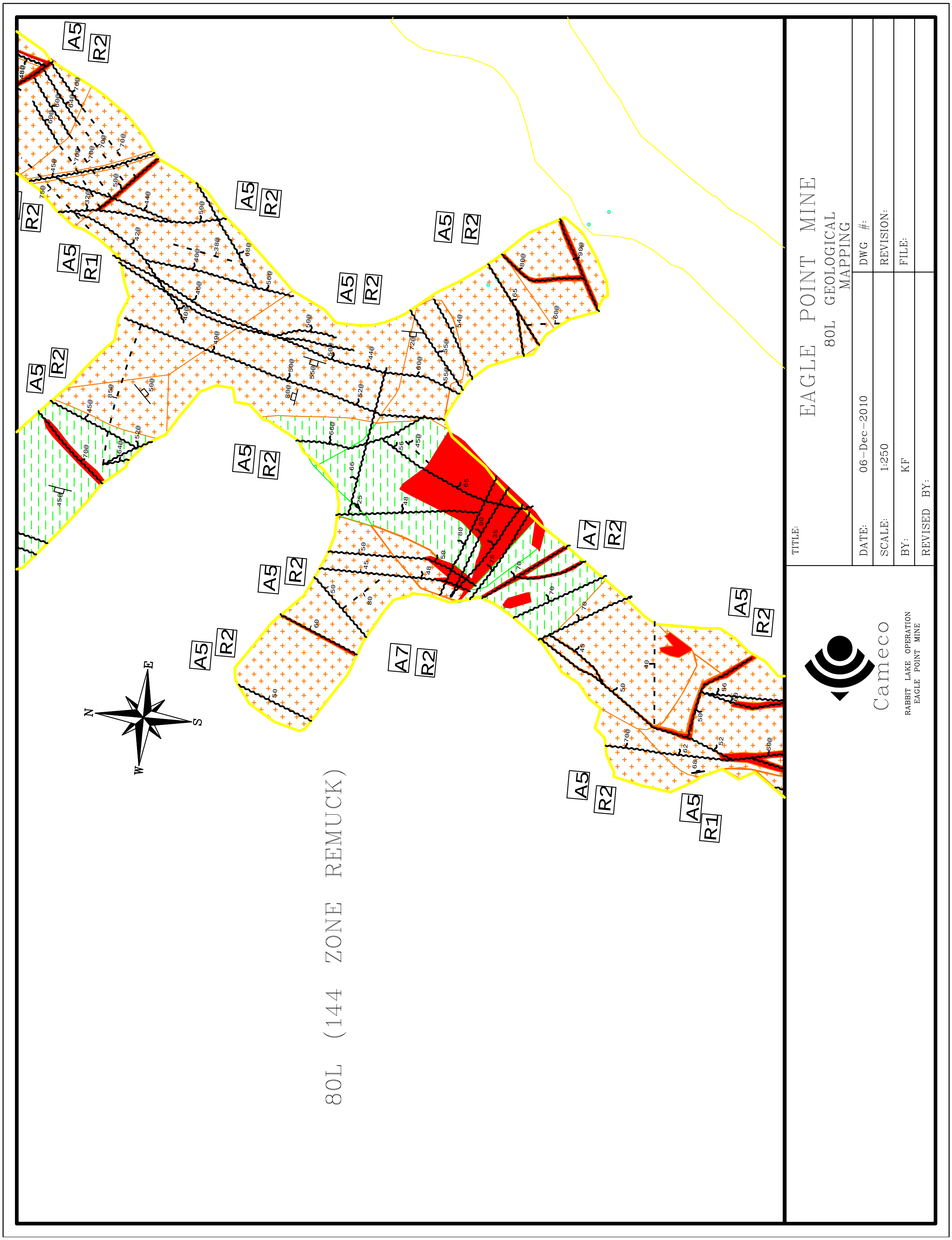


LOOKING AT 80L, (REMUCK . TYPICAL RMR FOR AREA APPROACHING 55% IN ROCK TO 35% WITHIN THE CLAY ZONES..

STOP #6: 80L, WASTE DEVELOPMENT (144 ZONE STH)

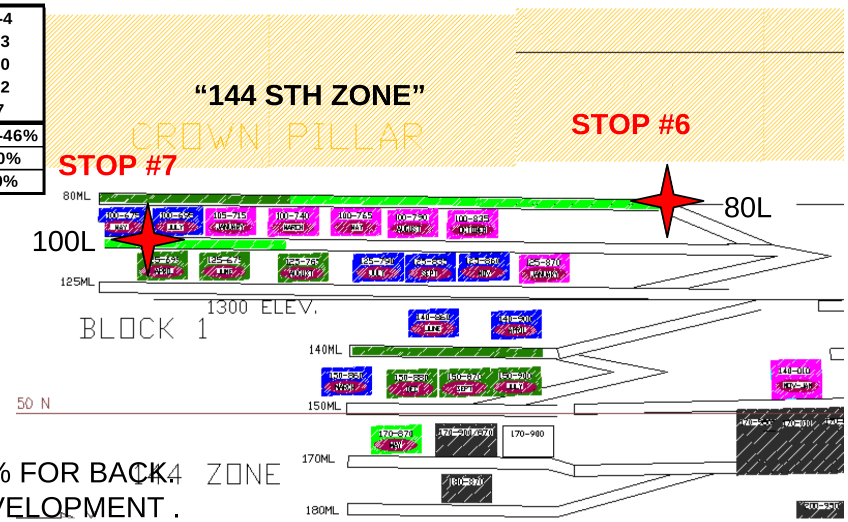
LOOKING WEST THIS IS THE PROBE/GROUT COVER FOR THE ACCESS. THE BACK IS 9m IN SPAN AND SUPPORT COMPRISED OF 5m(16ft) EXTENSION #7 REBAR + STD SUPPORT. (DIAMOND DRILL BAY SUPPORT). RMR ESTIMATED FROM 80L (REMUCK. GENERAL OBSERVATION IS WHERE CLAY ALTERED ZONES RMR APPROACHES 35% BUT GENERALLY 45-50%+.

29

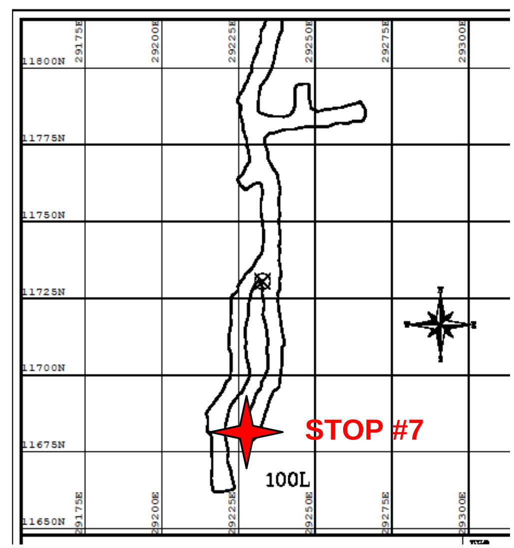

EAGLE POINT MINE
80L GEOLOGICAL MAPPING
TITLE:
DWG #:
REVISION:
FILE:
DATE: 06-Dec-2010
SCALE: 1:250
BY: KF
REVISED BY:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
80L (144 ZONE REMUCK)
N
E
S
W
A5
R2
A5
R1
A7
R2

RMR BACK/WALL ALT PEG

1) STRENGTH	50-10MPa-	7-4
2) RQD	75%-50%-	13
3) SPACING	50-300mm	10
4) CONDITON	SLT	12
5) GRNWTR	MOIST	7
STRUCTURE	RATING(WALL)	49%-46%
	FLAT	-10%
	DESIGN (BACK)	40%

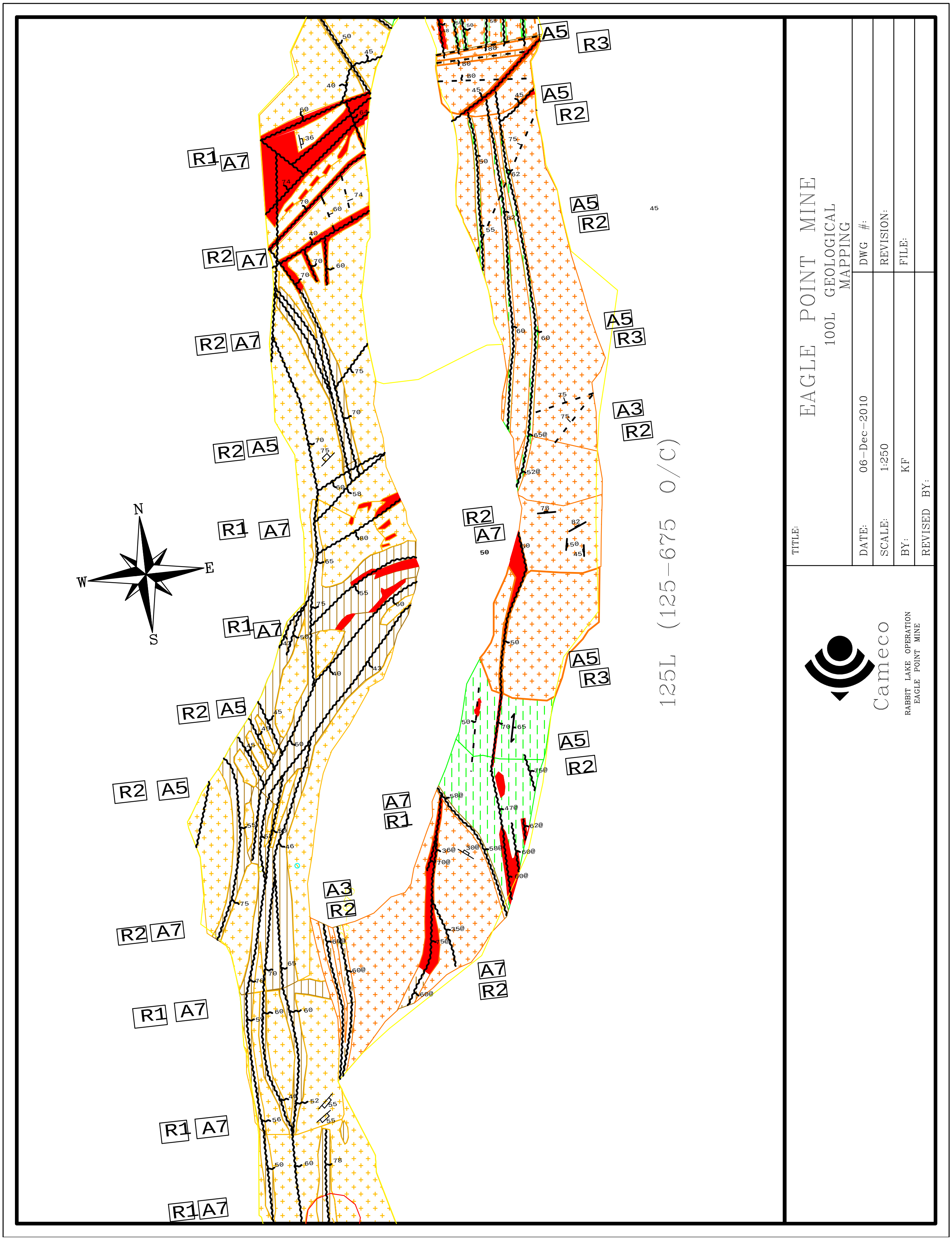


LOOKING SOUTH AT FACE. SPAN IS 5.5m WITH RMR OF 40% FOR BACK. STABLE WITH SUPPORT + SHOTCRETE IN PLACE. ORE DEVELOPMENT .

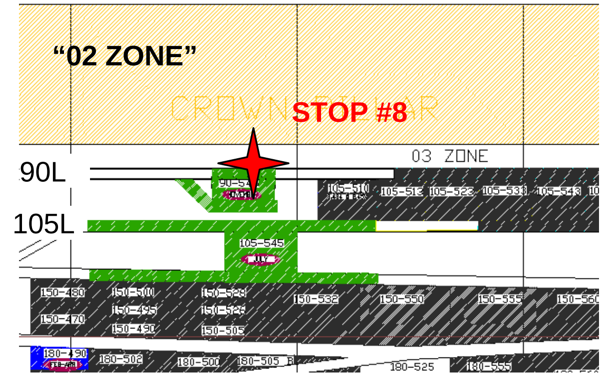


STOP #7: 100L, 144 ZONE U/C DEVELOPMENT (80-100)

LOOKING SOUTH AT FACE. THIS IS THE U/C FOR THE 144ZONE 80-100L STOPE. THE RMR IS 40% FOR BACK WITH SPAN OF 5.5m. STANDARD SUPPORT + SHOTCRETE WITH BOLTING THROUGH SHOTCRETE.


EAGLE POINT MINE
100L GEOLOGICAL MAPPING
TITLE:
DWG #:
REVISION:
FILE:
DATE: 06-Dec-2010
SCALE: 1:250
BY: KF
REVISED BY:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
125L (125-675 O/C)

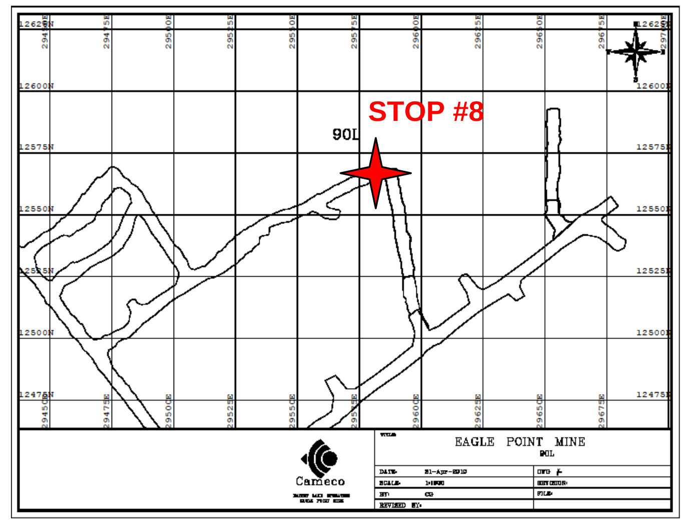
LOOKING NORTH AT FACE. RMR IS 56%+ IN BACK DUE TO FLAT JOINTS WITH SPAN OF 5m. STABLE WITH STD SUPPORT IN PLACE.



RMR BACK/WALL GNEISS

1) STRENGTH	100-200MPa	12
2) RQD	90%-75%	17
3) SPACING	50-300mm+	15
4) CONDITON	SLT	12
5) GRNWTR	DRY	10
	RATING(WALL)	66%
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	56%

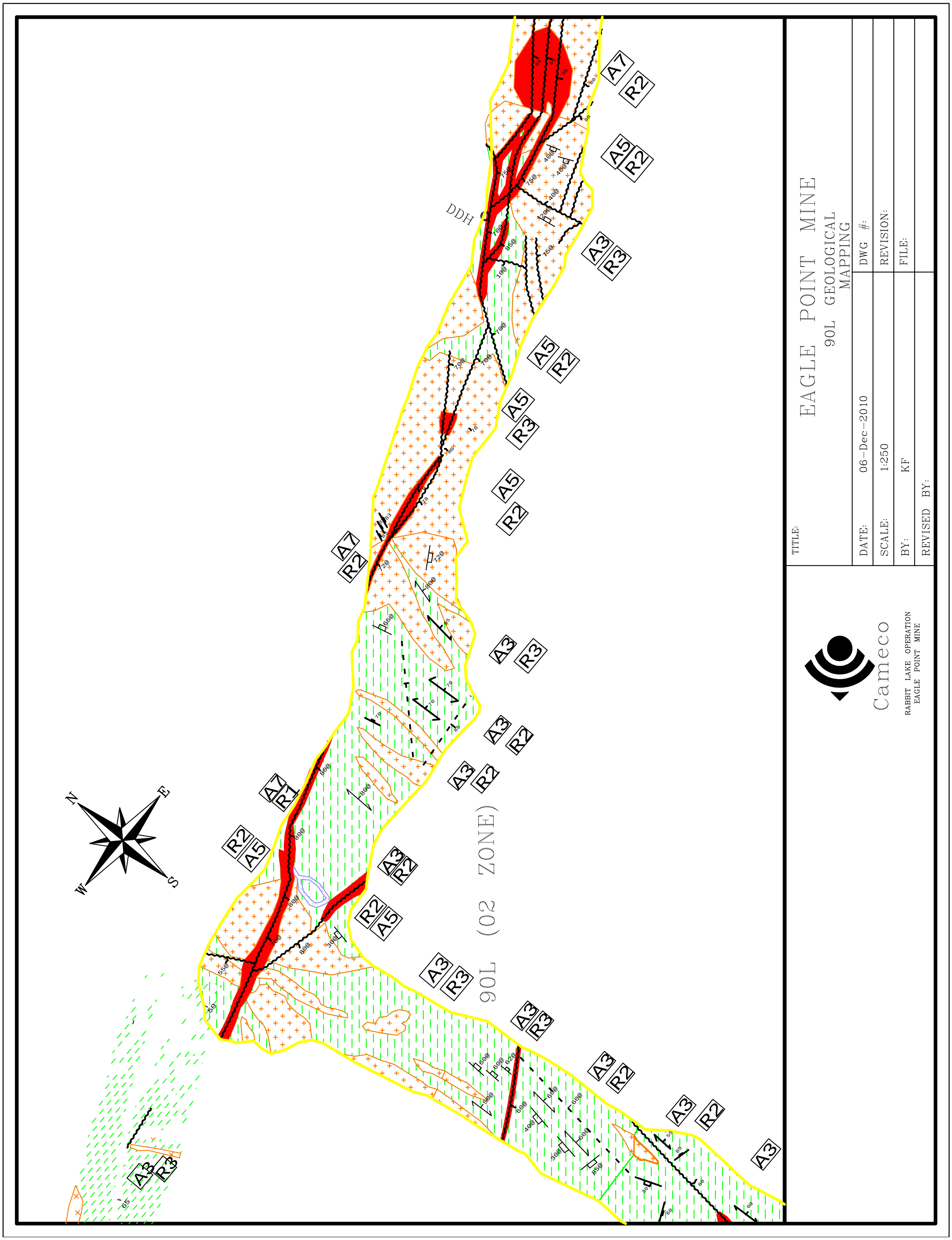
LOOKING AT EAST WALL(RIGHT) BLOCKY GROUND WITH VERTICAL RELEASE JOINTS COUPLED WITH FLAT JOINT SET WILL RESULT IN POTENTIAL BLOCKS.

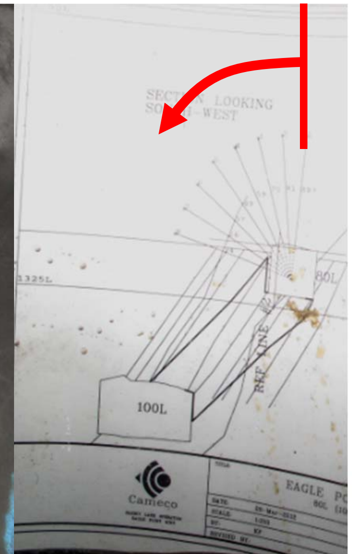


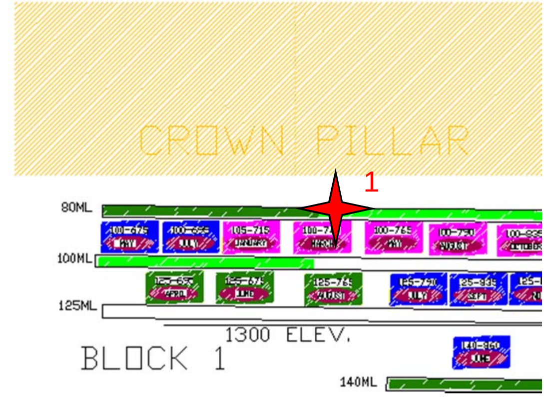
90L PLAN

STOP #8: 90L, "02 ZONE" ZONE U/C DEVELOPMENT (90-105) – ORE.

LOOKING NORTH AT FACE. THIS IS THE O/C FOR THE "02 ZONE" MINING STOPE BETWEEN 90L-105L. RMR IS 55%+ IN BACK WITH SPAN OF 5m. BACK IS BLOCKY BUT STD SUPPORT + SCREEN WILL CONFINE ANY POTENTIAL WEDGES. NOTE VERTICAL RELEASE PLANES COUPLED WITH FLAT JOINTS EXIST SO SUPPORT WILL BE CRITICAL TO ENSURE BACK STABILITY.TOP OF CROWN. SUPPORT COMPRISED OF STD SUPPORT.


EAGLE POINT MINE
90L GEOLOGICAL MAPPING
TITLE:
DATE: 06-Dec-2010
SCALE: 1:250
BY: KF
REVISED BY:
DWG #:
REVISION:
FILE:
Cameco
RABBIT LAKE OPERATION
EAGLE POINT MINE
90L (02 ZONE)
DDH

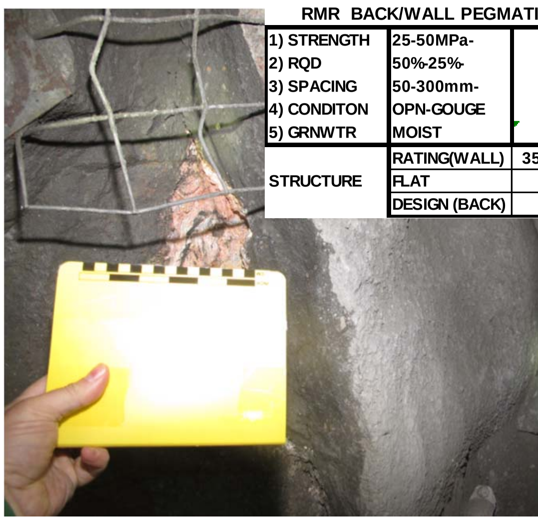





80L PLAN

LOOKING SOUTH TOWARDS FACE. CABLE BOLTING(LEFT) EAST WALL (HW)NE AND BACK. SINGLE CABLES 10m LONG ON FAN ~2m RING SPACING X 2.1m BURDEN. SPAN=4.6m-5.3m X 4.8m (H)

LOOKING SOUTH-WEST CABLING FROM CENTRE BACK TO HW. NOTE FW SIDE OF 80L DRIFT NOT CABLED. CROWN PILLAR CONCERN. CAVE BELOW. NOTE AREA WET AND ~50m FROM BOTTOM OF LAKE.



RMR BACK/WALL PEGMATITE

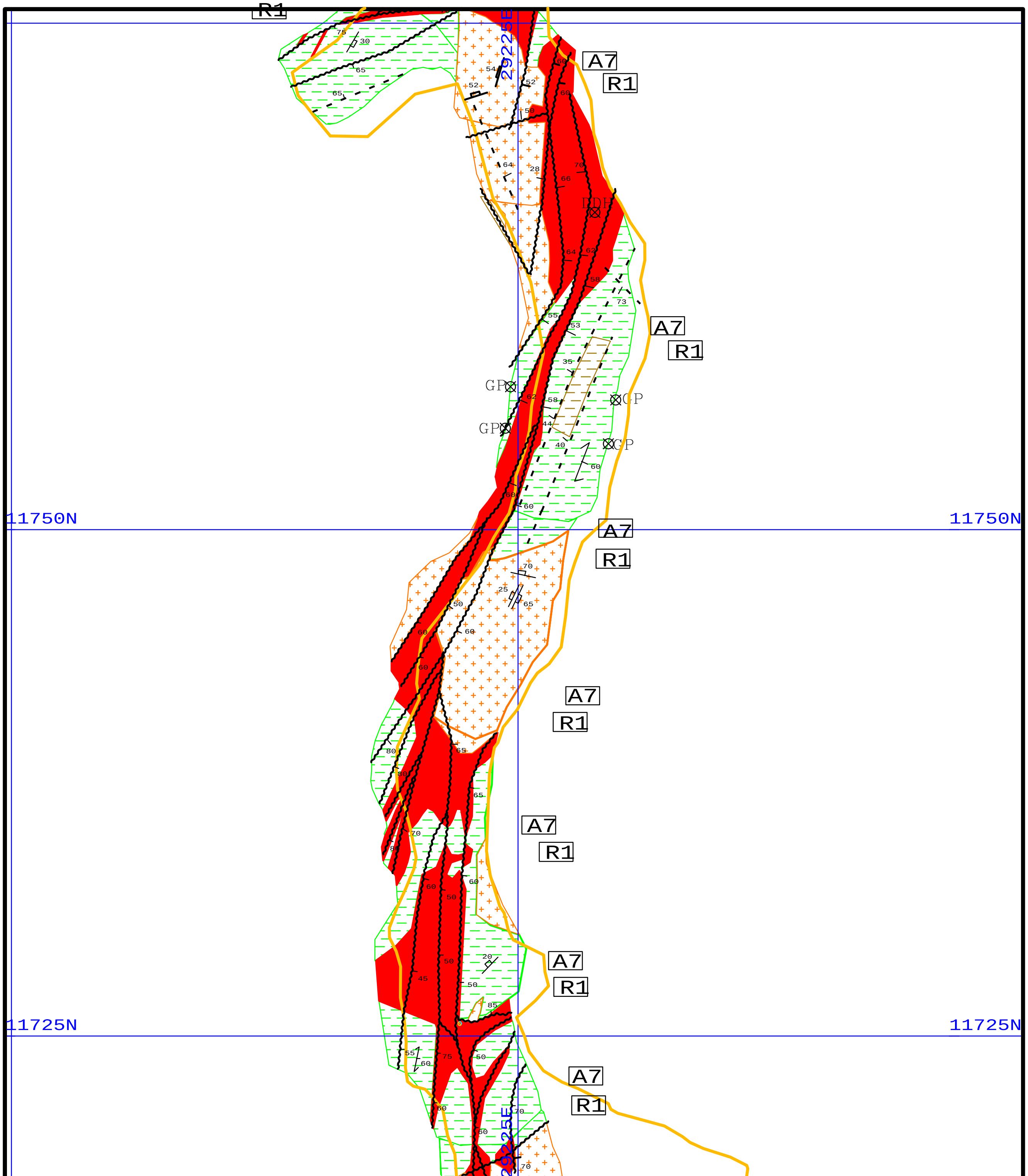
1) STRENGTH	25-50MPa-	4-2
2) RQD	50%-25%-	8-3
3) SPACING	50-300mm-	10-5
4) CONDITON	OPN-GOUGE	6-0
5) GRNWTR	MOIST	7
STRUCTURE	RATING(WALL)	35%-17%
	FLAT	
	DESIGN (BACK)	25%

RMR BACK/WALL ALT PEGMATITE

1) STRENGTH	50-100MPa	7
2) RQD	75%-50%	13
3) SPACING	50-300mm+	15
4) CONDITON	SLT	12
5) GRNWTR	DRY	7
STRUCTURE	RATING(WALL)	54%
	FLAT	'-10%
	DESIGN (BACK)	44%

SHOTCRETE CRACKING. GENERALLY OBSERVED RMR (KF).

STOP #1: 80L O/C 100-740 STOPE, “144 ZONE STH” (MINING 80L TO 100L)



Cameco

RABBIT LAKE OPERATION
EAGLE POINT MINE

TITLE:

EAGLE POINT MINE

STOP #1 – 80L
ROCK MX 2012

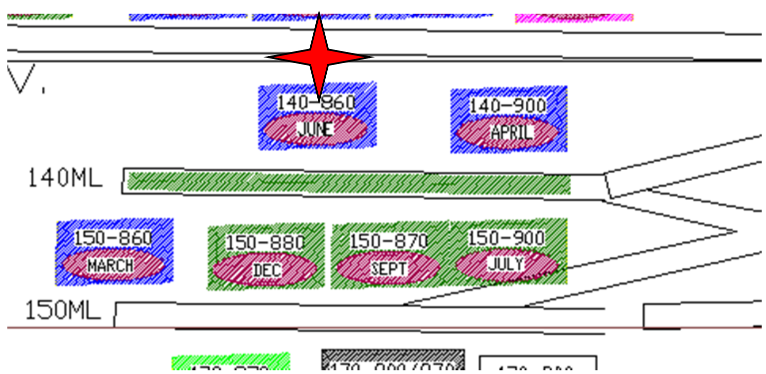
DATE:	07-Jun-2012	DWG #:	
SCALE:	1:250	REVISION:	
BY:	KF	FILE:	
REVISED BY:			

RMR BACK/WALL GNEISS

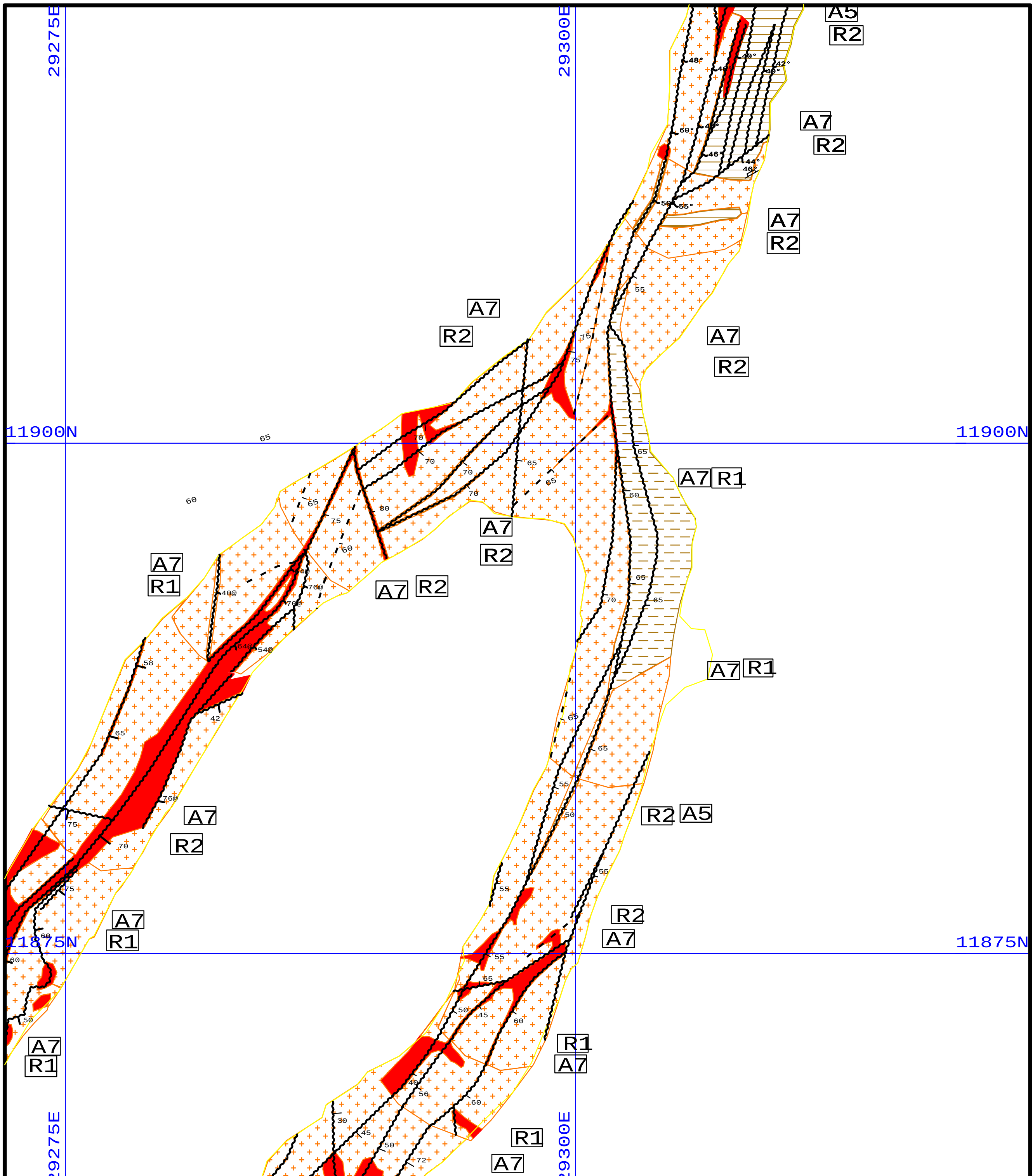
1) STRENGTH	50-100MPa	7
2) RQD	75%-50%	13
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT-SLT	20-12
5) GRNWTR	DRY	10
	RATING(WALL)	65%-57%
STRUCTURE	FLAT	MINOR
	DESIGN (BACK)	55%

Q'=(70/9) * (1.5/2) = 5.8 (RMR ~60%)

SPAN IS 4.6m W. STRIKE LENGTH OF STOPE ~25m. SUPPORT IS #6 REBAR WITH SHOTCRETE PLUS #6 GAUGE SCREEN ~6ft FROM FLOOR. ALL SHOTCRETED AREAS ARE SCREENED DUE TO POTENTIAL CRACKING AND PLACED FIRST DUE TO GAMMA. NO FIBRES IN SHOTCRETE.



STOP #2: 125L O/C, 140-860 STOPE (144 ZONE STH) MINED 125L TO 140L



Cameco

RABBIT LAKE OPERATION
EAGLE POINT MINE

TITLE:

EAGLE POINT MINE

125L
ROCK MX 2012

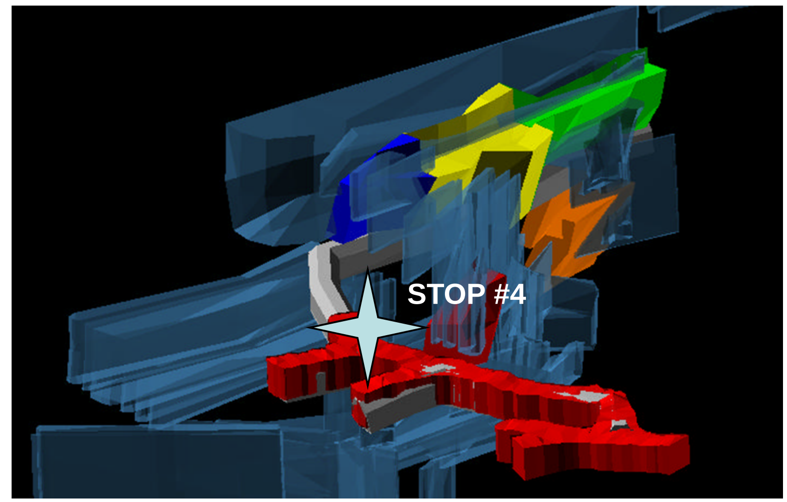
DATE:	07-Jun-2012	DWG #:	
SCALE:	1:250	REVISION:	
BY:	KF	FILE:	
REVISED BY:			

SPAN IS 5.3m (W). SUPPORT IS 2.4m LONG #6 REBAR ON 1.2m X 1.2m PATTERN. (EFFECTIVE LENGTH 2.3m). DEVELOPMENT IN WASTE SO NO SHOTCRETE AND 1.8m LONG SPLITSETS IN WALL.

"DEAD WEIGHT" WEDGE OBSERVED

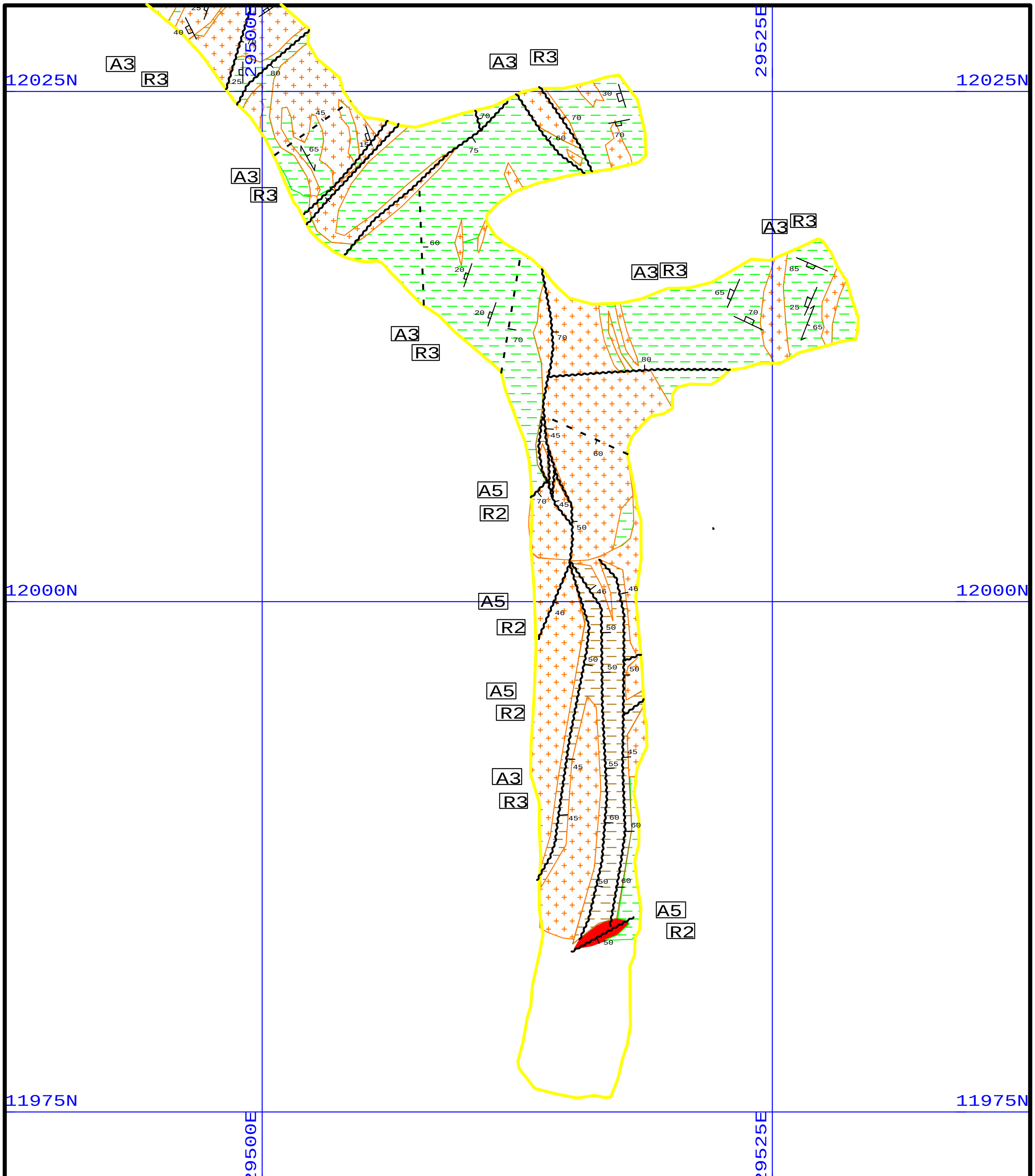
RMR BACK/WALL GNEISS

1) STRENGTH	100-200MPa	12
2) RQD	75%-50%	13
3) SPACING	50-300mm+	15-10
4) CONDITON	TIGHT	20
5) GRNWTR	DRY	10
STRUCTURE	RATING(WALL)	70%-65%
	FLAT	-10%
	DESIGN (BACK)	55%



DEVELOPMENT ACCESS TO 141 ZONE. SHOWNING VOIDS AND PROPOSED STOPES.

STOP #4: 151L DEVELOPMENT TOWARDS 141 ZONE (VOIDS)



TITLE: EAGLE POINT MINE

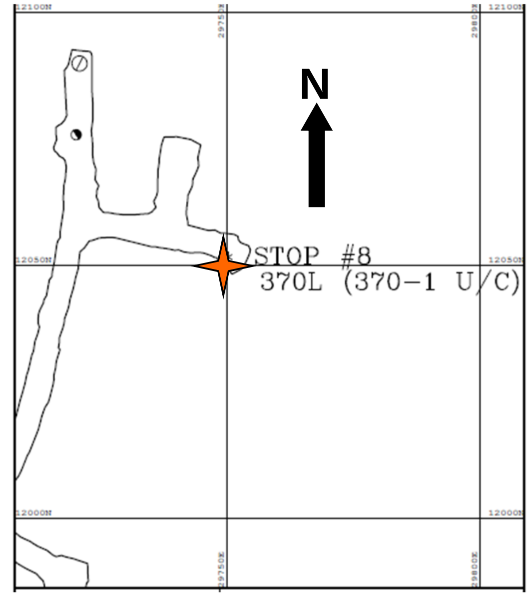
151L

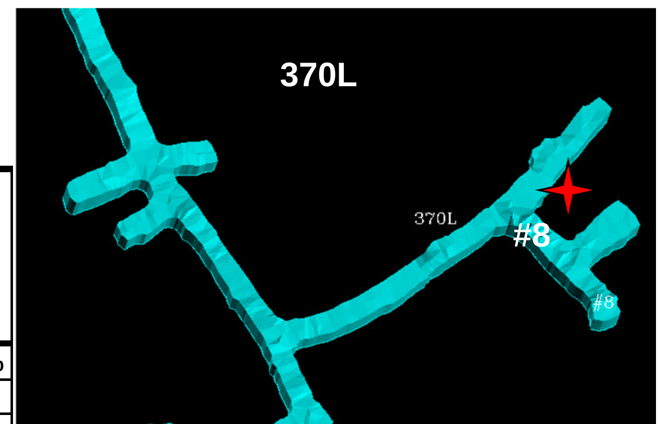
ROCK MX 2012

DATE:	07-Jun-2012	DWG #:	
SCALE:	1:250	REVISION:	
BY:	KF	FILE:	
REVISED BY:			

Cameco

RABBIT LAKE OPERATION
EAGLE POINT MINE



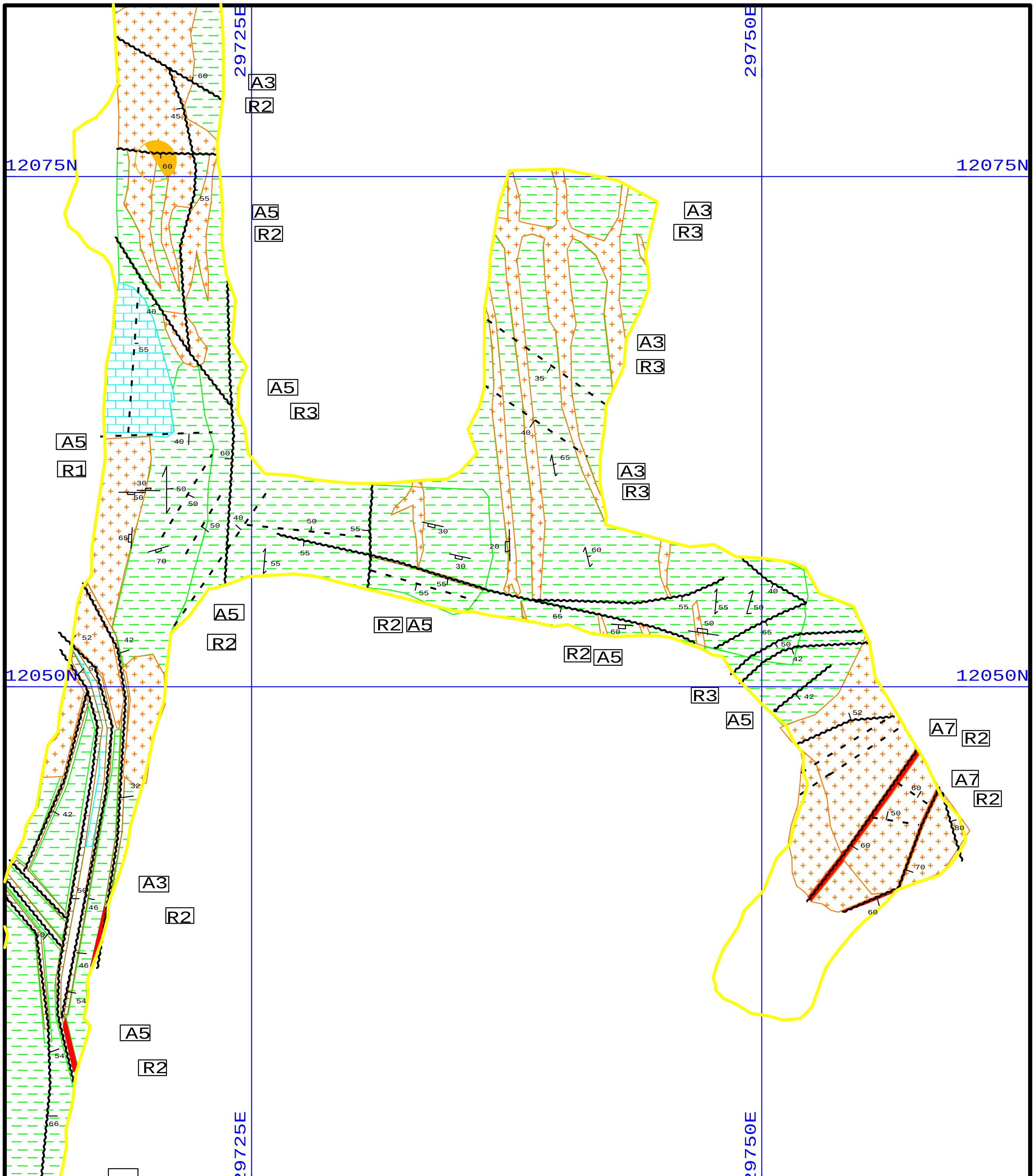


PROXIMITY OF THE ORE FW. HAVE A SPAN OF 5m WITH RMR OF 55%. CONCERN IS “DEAD WEIGHT” STRUCTURE, HOWEVER, SHOWN WITH SUPPORT IN PLACE WHICH IS 2.4m LONG #6 REBAR ON 1.2m X 1.2m PATTERN THE AREA IS STABLE. HAVE QUARTZ VUGS IN BACK (SOURCE OF WATER).

RMR BACK/WALL GNEISS

1) STRENGTH	100-200MPa	12
2) RQD	75%-50%	13
3) SPACING	50-300mm+	15
4) CONDITON	TIGHT-SLT	20-12
5) GRNWTR	DRY	10
	RATING(WALL)	72%-62%
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	55%

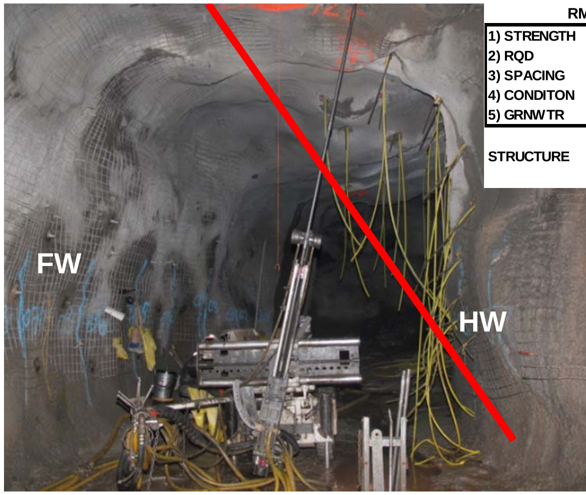
STOP #8: 370L, ORE ACCESS TO 370-1 STOPE U/C. 163 ZONE MINED FROM 370L-340L



Cameco

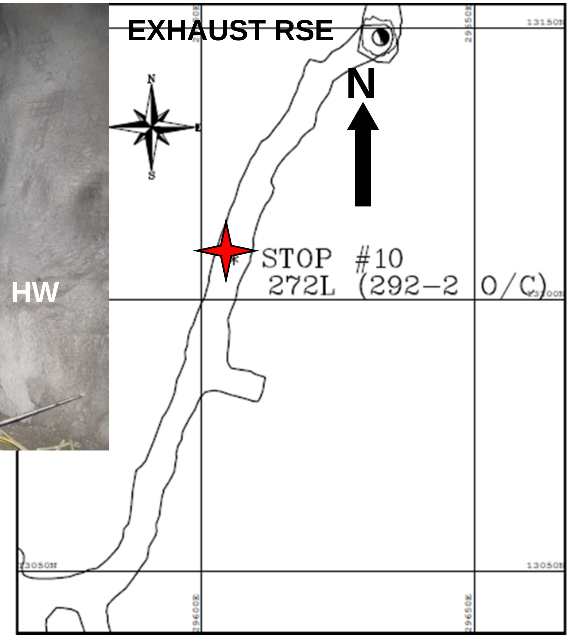
RABBIT LAKE OPERATION
EAGLE POINT MINE

TITLE: EAGLE POINT MINE 370L ROCK MX 2012	
DATE: 07-Jun-2012	DWG #:
SCALE: 1:250	REVISION:
BY: KF	FILE:
REVISED BY:	

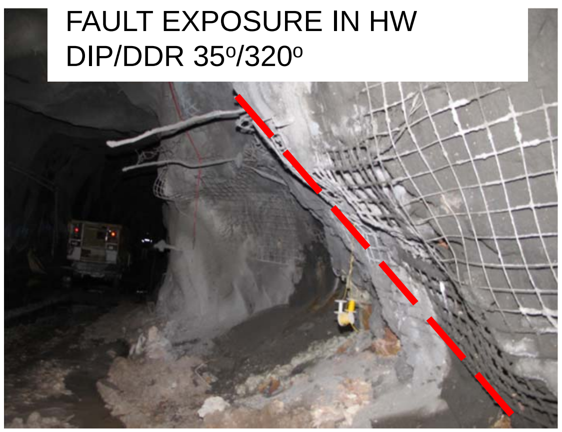


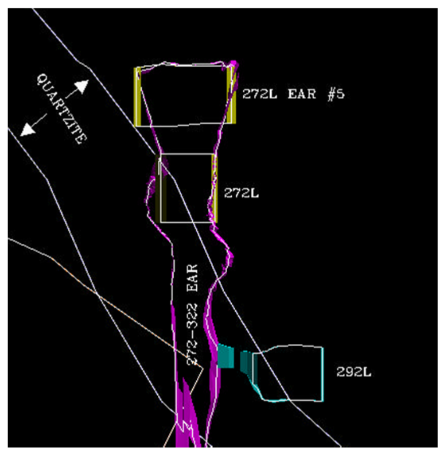
RMR FW GNEISS

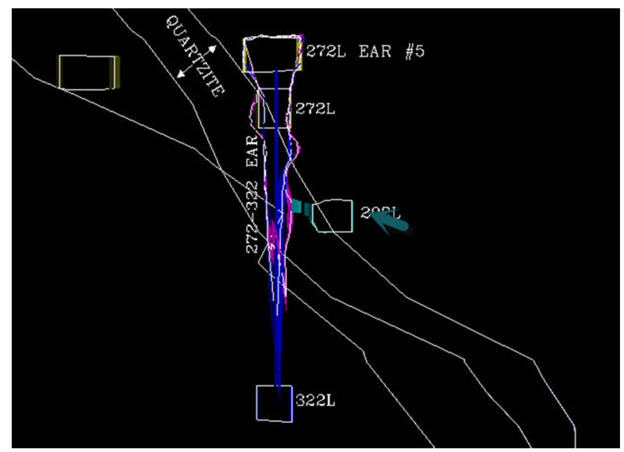
1) STRENGTH	100-200MPa	12
2) RQD	75%-50%	13
3) SPACING	50-300mm+	15-10
4) CONDITON	TIGHT	20-12
5) GRNWTR	DRY	7
	RATING(WALL)	67%-59%
STRUCTURE	FLAT	-10%
	DESIGN (BACK)	55%



LOOKING NE. RMR OF ORE IS 25%-30%. STANDARD 2.4m LONG #7 REBAR ON 1.2 X 1.2m PATTERN, SPAN IS 5.6m WITH LEFT WALL(NW) WALL IS FW HAVING RMR OF 55%. BACK AND HW HAS RMR OF 25%-30% WITH MOST CRACKING OBSERVED ON RIGHT WALL(HW).

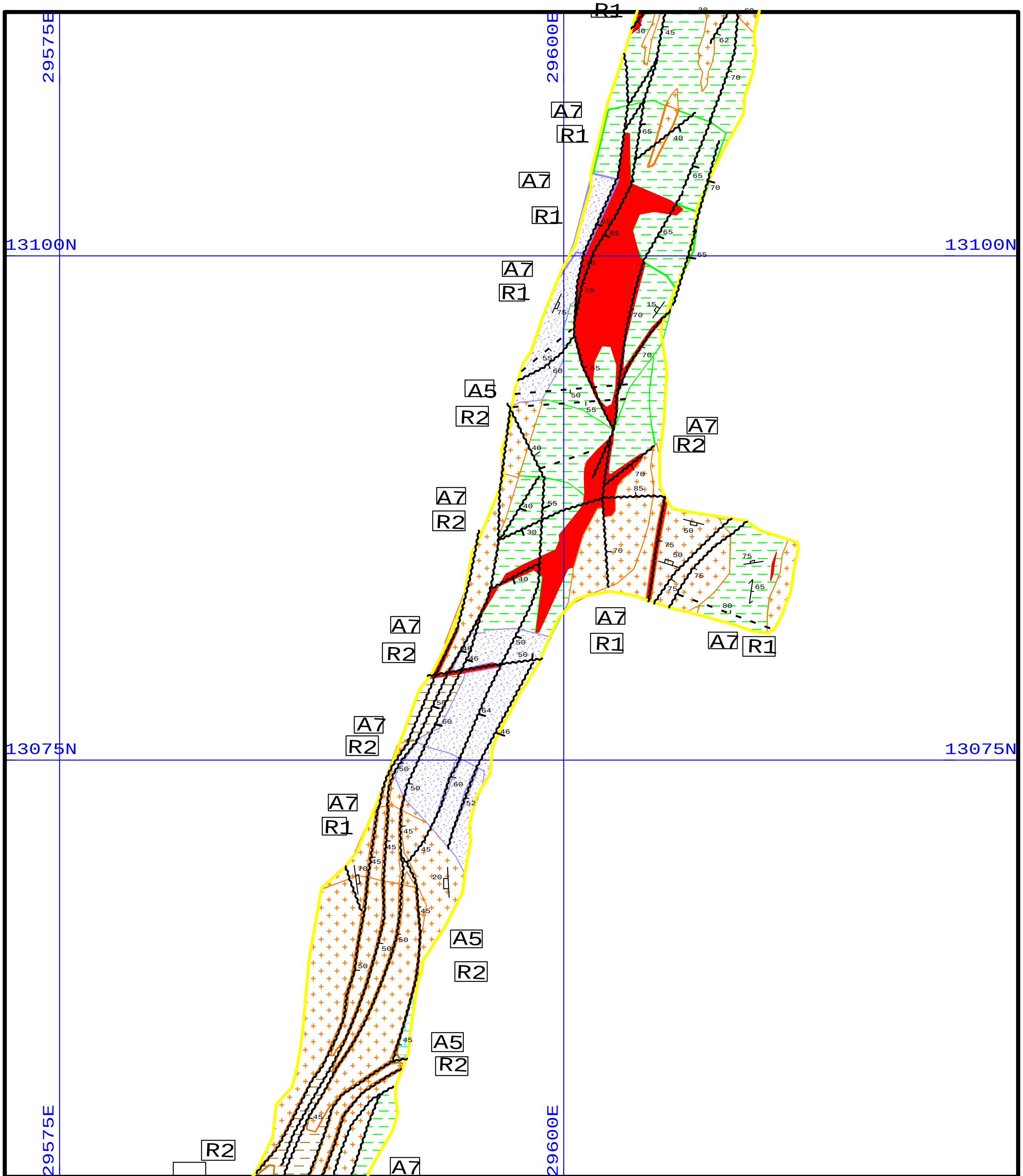






HAD ~4.5m OF FAULT CUTTING THROUGH 4m DIAMETER RAISE THAT WAS BLASTED FROM 322L-272L. FAULT COMPRISED OF CLAY/GRAPHITE/WATER (COLLINS BAY FAULT?) WITH ADJACENT ROCK AND MAJORITY OF EXHAUST RAISE BEING 60% RMR.

STOP #10: 292L, 292-2 STOPE O/C. 02 NEXT FW ZONE MINED FROM 310L-292L



Cameco

RABBIT LAKE OPERATION
EAGLE POINT MINE

TITLE:

EAGLE POINT MINE

272L 02NFW
ROCK MX 2012

DATE:	07-Jun-2012	DWG #:	
SCALE:	1:250	REVISION:	
BY:	KF	FILE:	
REVISED BY:			

Appendix D

Site Mapping Data Sheets

(A)

ROCK MASS CLASSIFICATION SHEET - drift mapping

LOCATION:
collected by:
date:

Rock type Gneiss with pegmatite		Strength (Mpa) - Schmidt hammer 21 MPa		
Jointing*	set1	set 2	set 3	set 4
Dip/dip azimuth	80/80	90/90	60/260	
Av. Spacing(m)	S1= 0.75	S2= 0.3	S3= 0.3	S4=

Joints/m3 Jv = 1/S1+1/S2+1/S3+1/S4= 8

NGI classification
RQD= 115-3.3*Jv = 89 %
Jn = 9
Jr = 1.5
Ja = 6 – altered
Is water present ? trace

Q' = (RQD/Jn)*(Jr/Ja) = (89/9)*(1.5/6) = 2.5

*) any weakness plane: open fracture, fracture filled with clay or graphite, foliation planes, etc.

(B)

Rock Mass Classification Recording Sheet

Site / Span:	Collected by:	Date:
382L Ramp	K. Forster	31-JULY-10

Jnt Alteration / Infilling / Thickness R / A

Chlorite
1 mm thick.

Rock type	Strength	Jnts/m3
Granitic Dome.	R3 or greater	16 J/m^3.

Structure Data	Set A	Set B	Set C	Set D
Strike/Dip	80°	20°	85°	
Average Spacing (m)	0.10	1 m	0.20	
JRC	Rough	~~Wavy~~ Rough	Rough.	
Amp/m	Wavy	Wavy	Wavy	
Groundwater Condition	Damp.			

RMR Classification	Value	RMR$_{76}$	Q' Classification	
RQD 95	20		RQD	95
Intact Rock Strength	12		Joint Set No.	12
Joint Spacing	12		Joint Roughness No.	3
Joint Condition	20		Joint Alteration	4
Groundwater	15		Joint Water Reduction	1.0
			Stress Reduction (1.0)	1.0
RMR'	79.		Tunnelling Quality Q'	~~10.5~~ 5.9

Other Notes:

3+ random joint sets.

pegmatoid intrusions.

c

Rock Mass Classification Recording Sheet

Site / Span:	Collected by:	Date:
3702-1632	K. FORSTER	2-JULY-2011

Jnt Alteration / Infilling / Thickness R / A

minimal joint alteration (except for shear) (trace clay & trace chlorite)

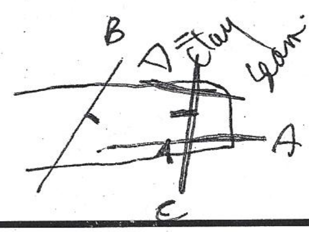
Rock type	Strength	Jnts/m3
Peg & Gneiss	R3 = 100-200 MPa	

Structure Data	Set A	Set B	Set C	Set D
~~Strike/Dip~~ Dip / Dip Dir	80/320	80/330	40/120	85/060
Average Spacing (m)	0.3-0.5	0.3-0.5m	0.3-0.5m	5cm-thick
JRC	Smooth	Smooth	Smooth	clay/gouge
Amp/m	Wavy.	Wavy	Wavy.	
Groundwater Condition	Dry			

RMR Classification	Value	RMR_{76}	Q' Classification	
RQD	90	20	RQD	90
Intact Rock Strength		12	Joint Set No.	12
Joint Spacing		10	Joint Roughness No.	2
Joint Condition		20	Joint Alteration	2
Groundwater		15	Joint Water Reduction	1.0
			Stress Reduction (1.0)	1.0
RMR'		77	Tunnelling Quality Q'	7.5

Other Notes:

Drift Dir: 120°



5

Rock Mass Classification Recording Sheet

Site / Span: 232X - (02 NExt plug #1) 272 EAR#5	Collected by: K. Forster	Date: 24-Nov-11

Jnt Alteration / Infilling / Thickness R / A : A3 / R3

Drift has been in place for several years, joint set alteration difficult to assess, but pieces removed from wall appear to be clean to minimal infilling

Rock type	Strength	Jnts/m3
BQFG (Biotite-Quartz-Feldspar Gneiss)	100-200 MPa	15

Structure Data	Set A	Set B	Set C	Set D
Strike/Dip	190/40E	180/50W	280/85N	
Average Spacing (m)	0.1	0.3-0.5	0.3-0.5	
JRC	sl. rough	sl. rough	sl. rough	
Amp/m	wavy	wavy	wavy	
Groundwater Condition	dry	dry	dry	

RMR Classification	Value	RMR_{76}	Q' Classification	
RQD	90	17	RQD	90
Intact Rock Strength		12	Joint Set No.	9
Joint Spacing		10	Joint Roughness No.	3
Joint Condition		19	Joint Alteration	2
Groundwater		15	Joint Water Reduction	1
			Stress Reduction (1.0)	1.0
RMR'		73	Tunnelling Quality Q'	15

Other Notes: Joint set "A" is the predominant foliation
Area will require rehab before construction of plug (scaling, bolting and screening)

Ⓔ

Rock Mass Classification Recording Sheet

Site:	Collected by:	Date:
322L B/H 4	K. FORSTER	23-NOV-11

Description of sample area:
Location for O2 Next Plug #4

Rock type	Strength	Jnts/m3
BQFG + EXPO	50 - 100 MPa	

Structure Data	Set A	Set B	Set C	Set D
Strike/Dip	060/30°	050/030	350/060	
Average Spacing (m)	0.5m	0.1m	0.5m	
JRC	Rough	Rough	Rough	
Amp/m	Wavy	Wavy	Wavy	
Jnt Alteration / Infilling / Thickness	N/A	N/A	N/A	

RMR Classification	RMR_{76}		Q' Classification	
RQD			RQD	
Intact Rock Strength			Joint Set No.	
Joint Spacing			Joint Roughness No.	
Joint Condition			Joint Alteration	
Groundwater			Groundwater	
			Stress Reduction (1.0)	
RMR'			Tunnelling Quality Q'	

Other Notes:

Ⓕ

Rock Mass Classification Recording Sheet

Site / Span: 272X Concrete Plug	Collected by: K. Pawliuk and K. Forster	Date: 11-May-2011

Jnt Alteration / Infilling / Thickness R / A : A3 / R2
Moderate alteration on joint sets, some foliation, clay/gouge seam

Rock type	Strength	Jnts/m3
FXPO (Feldspar Porphyry)	100 Mpa	9

Structure Data	Set A	Set B	Set C	Set D
Strike/Dip				
Average Spacing (m)	0.2	0.5	0.5	
JRC	sl. rough	sl. rough	sl. rough	
Amp/m	wavy	wavy	wavy	
Groundwater Condition	wet	wet	wet	

RMR Classification	Value	RMR_{76}	Q' Classification	
RQD	90	17	RQD	90
Intact Rock Strength		10	Joint Set No.	9
Joint Spacing		10	Joint Roughness No.	1.5
Joint Condition		10	Joint Alteration	4
Groundwater		7	Joint Water Reduction	1
			Stress Reduction (1.0)	1.0
RMR'		54	Tunnelling Quality Q'	3.75

Other Notes:

6

Rock Mass Classification Recording Sheet

Site / Span: 232L Concrete Plug (up-ramp)	Collected by: K. Pawliuk and K. Forster	Date: 11-May-2011

Jnt Alteration / Infilling / Thickness R / A : A3 / R3
Little alteration on joint sets, tight joints/foliation, some shear zones in the area

Rock type	Strength	Jnts/m3
BQFG (Biotite-Quartz-Feldspar Gneiss)	100-200 Mpa	14

Structure Data	Set A	Set B	Set C	Set D
Strike/Dip				
Average Spacing (m)	0.1	0.5	0.5	
JRC	sl. rough	sl. rough	sl. rough	
Amp/m	wavy	wavy	wavy	
Groundwater Condition	dry	dry	dry	

RMR Classification	Value	RMR_{76}	Q' Classification	
RQD	90	17	RQD	90
Intact Rock Strength		12	Joint Set No.	9
Joint Spacing		10	Joint Roughness No.	1.5
Joint Condition		18	Joint Alteration	2
Groundwater		10	Joint Water Reduction	1
			Stress Reduction (1.0)	1.0
RMR'		67	Tunnelling Quality Q'	7.5

Other Notes:

(H)

Rock Mass Classification Recording Sheet

Site / Span: 232L Concrete Plug (down-ramp)	Collected by: K. Pawliuk and K. Forster	Date: 11-May-2011

Jnt Alteration / Infilling / Thickness R / A : A3 / R3
Little alteration on joint sets, tight joints/foliation, some shear zones in the area

Rock type	Strength	Jnts/m3
BQFG (Biotite-Quartz-Feldspar Gneiss), some pegmatoid	100-200 Mpa	14

Structure Data	Set A	Set B	Set C	Set D
Strike/Dip				
Average Spacing (m)	0.1	0.5	0.5	
JRC	sl. rough	sl. rough	sl. rough	
Amp/m	wavy	wavy	wavy	
Groundwater Condition	dry	dry	dry	

RMR Classification	Value	RMR_{76}	Q' Classification	
RQD	90	17	RQD	90
Intact Rock Strength		12	Joint Set No.	9
Joint Spacing		10	Joint Roughness No.	1.5
Joint Condition		18	Joint Alteration	2
Groundwater		10	Joint Water Reduction	1
			Stress Reduction (1.0)	1.0
RMR'		67	Tunnelling Quality Q'	7.5

Other Notes:

Ⓘ

Strike of [illegible] 85°

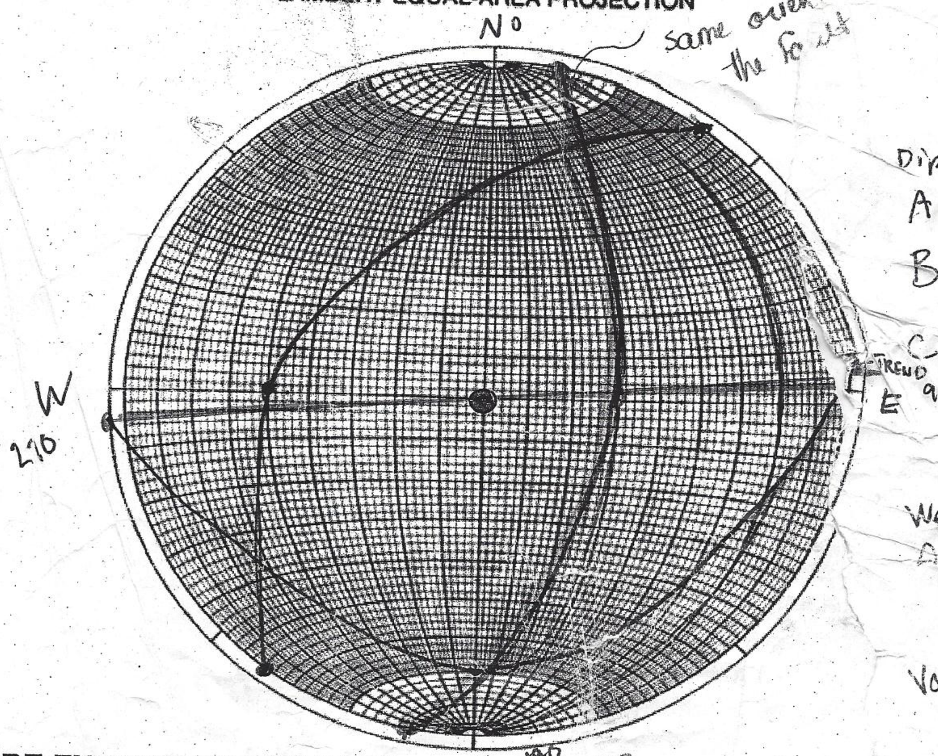
INTERSECTION ANALYSIS DATA SHEET

Intersection Location: 272x Remuck
Analysis By: K. COATES Murray DUNNE
Date: 3 Nov 10

	JOINT SET			
	Fault A	B	C	D
Average Strike	10°	85	215°	
Average Dip	60°	20°	40°	
Rock Type	Pegmatite			
Rock Strength	150 mpa			
Joint Spacing	10 cm	30cm-50cm	50 cm	
Joint Alteration	Clean	[illegible]	clay	
Joint Roughness	6-8	8-10	6-8	
Joint Amplitude	2cm	< 1cm	10cm	

- Treat a fault as a joint set if it crosses the intersection

LAMBERT EQUAL-AREA PROJECTION



-90°

Dip Direction
A 100°
B 175°
C 305°

Wedge Apex Heigh
0.76m
Volume 1.16 m3

ARE EXTENSION REBAR REQUIRED? Y / N
IF SO, HOW MANY? HR of Intersection only 6m

the area was foliated with a fault running the same direction as the foliation

Ⓙ

59°
50°

INTERSECTION ANALYSIS DATA SHEET

Intersection Location: 302 TTA
Analysis By: K. COATES
Date:

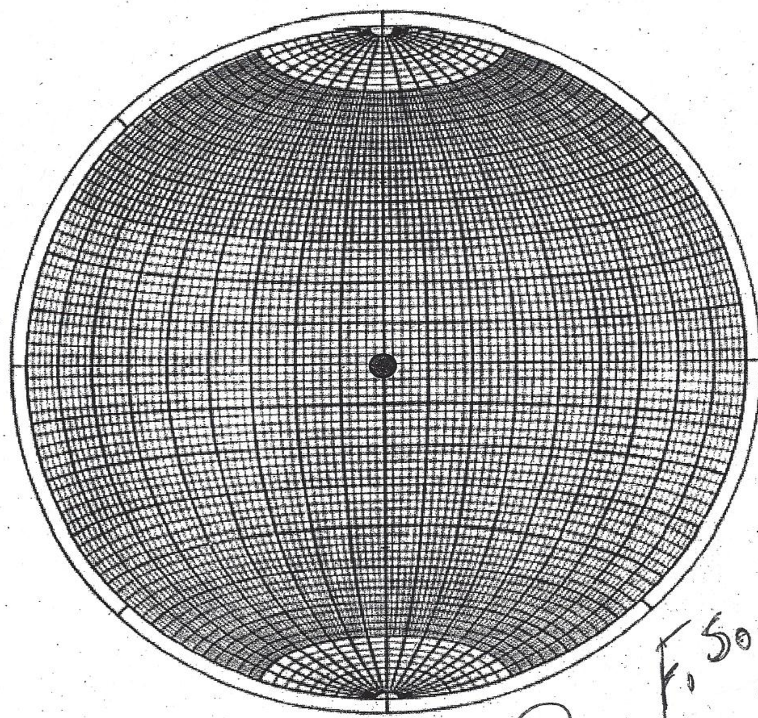
	JOINT SET			
	A	**B**	**C**	**D**
Average Strike	120°	260°	70°	40-70°
Average Dip	55°	60-75°	20-40°	80
Rock Type	PEG	REG	PEG	PEG
Rock Strength	R2-R3	—	—	—
Joint Spacing	0,3-0,5	>1m	0.3-0.5m	20cm
Joint Alteration	none	none		
Joint Roughness	Medium	Medium	—	—
Joint Amplitude	2cm	1cm	—	—

Strongest 2 weak

- Treat a fault as a joint set if it crosses the intersection

LAMBERT EQUAL-AREA PROJECTION

Strike of dip t
140
-3°



F.S. 2.34 w/ Standard Bolting. KC 10Feb11

ARE EXTENSION REBAR REQUIRED? Y / N
IF SO, HOW MANY?

K

INTERSECTION ANALYSIS DATA SHEET

Intersection Location: 302L Ramp & 302L Remuck.
Analysis By: KARINA FORSTER.
Date: 18-JAN-2011

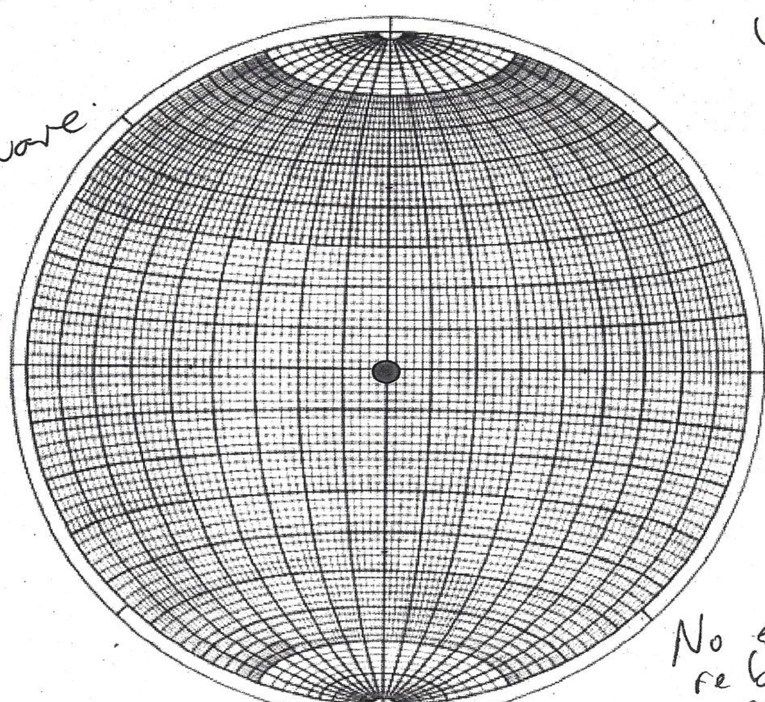
	JOINT SET			
	A	**B**	**C**	**D**
Average Dip Dir	140°	290°	040°	
Average Dip	20°-70°	40°-50°	10°-80°	
Rock Type	PEG			
Rock Strength	R2-R3			
Joint Spacing	0.5m	0.3m	1m.	
Joint Alteration	Trace Chlorite			
Joint Roughness	Rough			
Joint Amplitude	Wavy			

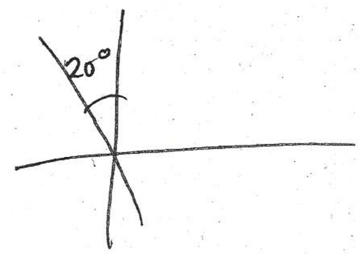
Drift
12°31'38"
-14%
Size:
5.2m W
x4.9m H

- Treat a fault as a joint set if it crosses the intersection

LAMBERT EQUAL-AREA PROJECTION * Dip angles from vertical.

Average Block size = Microwave.





No extension rebar required.
KF

ARE EXTENSION REBAR REQUIRED? Y N
IF SO, HOW MANY?

~~2/3 extension rebar.~~

KF

L

INTERSECTION ANALYSIS DATA SHEET

Intersection Location: 272 EAR #4 ACCESS & 272 EAR#4 REMUCK #1
Analysis By: KARINA FORSTER
Date: 18-JAN-2011

	JOINT SET			
	A	B	C	D
Average Dip Dir	080	300°	10°	
Average Dip	20 deg 70°	30 deg 60°	20° deg 70°	
Rock Type	Granitic Dome			
Rock Strength	R3			
Joint Spacing	0.3m	0.5m	1m	
Joint Alteration	Trace chlorite	trace Chl.	trace Chlo	
Joint Roughness	Smooth	Smooth	Smooth	
Joint Amplitude	Wavy	Wavy.	Wavy	

Drift 005° +3%

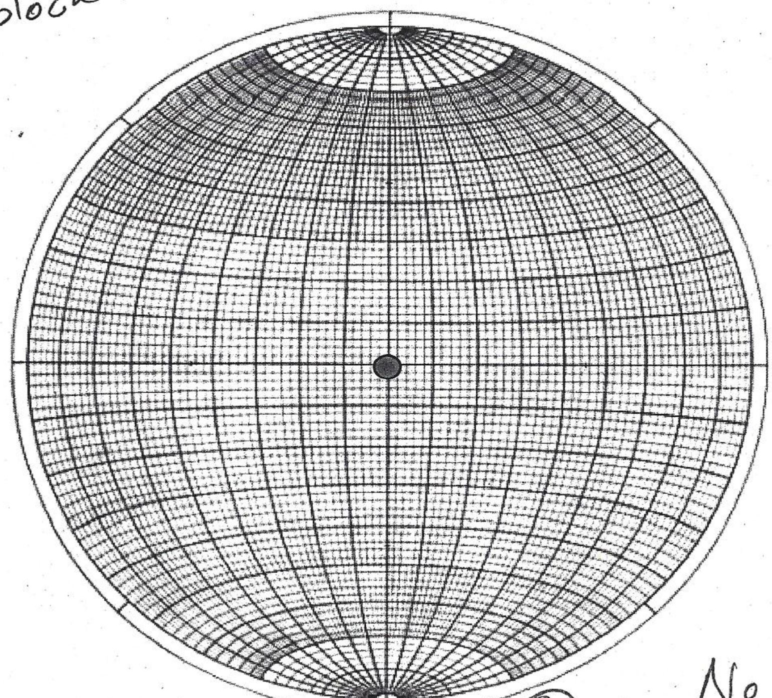
Size: 5.2W x 4.9H

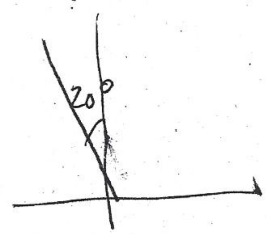
- Treat a fault as a joint set if it crosses the intersection

dip from vertical!

LAMBERT EQUAL-AREA PROJECTION

Average block size 27" TV.





ARE EXTENSION REBAR REQUIRED? Y / N
IF SO, HOW MANY? ______

No extension rebar required.
KF.

Appendix E

Case Histories – Stope Reconciliations

STOPE	100-015	
Strike Length	30.5	
HW Exposed Height	25.1	
Hydraulic Radius	6.9	
Average HW Dip	52.7	°
Percentage of HW that is:		
Gneissic	79	%
Pegmatoidal	21	%
Other	0	%

100-015						Sutton's Q' from A/R values					
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
Gneiss	A7/R1	12	20	240	4	48	0.75	9	10	120	
	A7/R2	7.5	60	450	6	45	1.5	11.25	6	45	
	A5/R2	11	75	825	7.5	82.5	1.5	16.5	6	66	
TOTAL:		30.5		1515		175.5		36.75		231	
				49.7		5.8		1.2		7.6	1.4
UNDERCUT											
Gneiss	A5/R2	12.7	75	952.5	7.5	95.25	1.5	19.05	6	76.2	
	A5/R3	1	100	100	9	9	1.5	1.5	2	2	
	A3/R3	3.8	100	380	9	34.2	1.5	5.7	1.5	5.7	
Pegmatoid	A5/R2	6	75	450	7.5	45	1.5	9	6	36	
	A5/R3	7	100	700	9	63	2.3	16.1	2	14	
TOTAL:		30.5		2582.5		246.45		51.35		133.9	
				84.7		8.1		1.7		4.4	4.0
									AVERAGE:		2.7

DDH's	RQD in HW:
2193	87

Q' Weighted	RQD	Jn	Jr	Ja
2.6	73.8	6.9	1.4	6.0

A	B	C	N'
1	0.2	4.4	2.3

100-015

Parameter Based Q' from A/R values

OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
Gneiss	A7/R1	12	75	900	9	108	0.75	9	4	48	
	A7/R2	7.5	75	562.5	9	67.5	1.5	11.25	4	30	
	A5/R2	11	75	825	9	99	1.5	16.5	3	33	
TOTAL:		30.5		2287.5		274.5		36.75		111	
				75		9		1.2		3.6	2.8
UNDERCUT											
Gneiss	A5/R2	12.7	75	952.5	9	114.3	1.5	19.05	3	38.1	
	A5/R3	1	75	75	9	9	1.5	1.5	2	2	
	A3/R3	3.8	75	285	9	34.2	1.5	5.7	1.5	5.7	
Pegmatoid	A5/R2	6	60	360	7.5	45	1.5	9	4	24	
	A5/R3	7	75	525	9	63	2.3	16.1	2	14	
TOTAL:		30.5		2197.5		265.5		51.35		83.8	
				72.0		8.7		1.7		2.7	5.1
									AVERAGE:		3.9

DDH's	RQD in HW:
2193	87

Q' Weighted	RQD	Jn	Jr	Ja
4.0	78.0	8.9	1.4	3.2

A	B	C	N'
1	0.2	4.4	3.5

100-015

RMR = 9 ln Q' + 44

OVERCUT		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
Gneiss	A7/R1	12	69	828	3.6	43.2
	A7/R2	7.5	73	547.5	5	37.5
	A5/R2	11	68	748	8.3	91.3
TOTAL:		30.5		2123.5		172
				69.6		5.6
UNDERCUT						
Gneiss	A5/R2	12.7	68	863.6	8.3	105.41
	A5/R3	1	73	73	14	14
	A3/R3	3.8	77	292.6	15.7	59.66
Pegmatoid	A5/R2	6	63	378	2.5	15
	A5/R3	7	79	553	18	126
TOTAL:		30.5		2160.2		320.07
				70.8		10.5

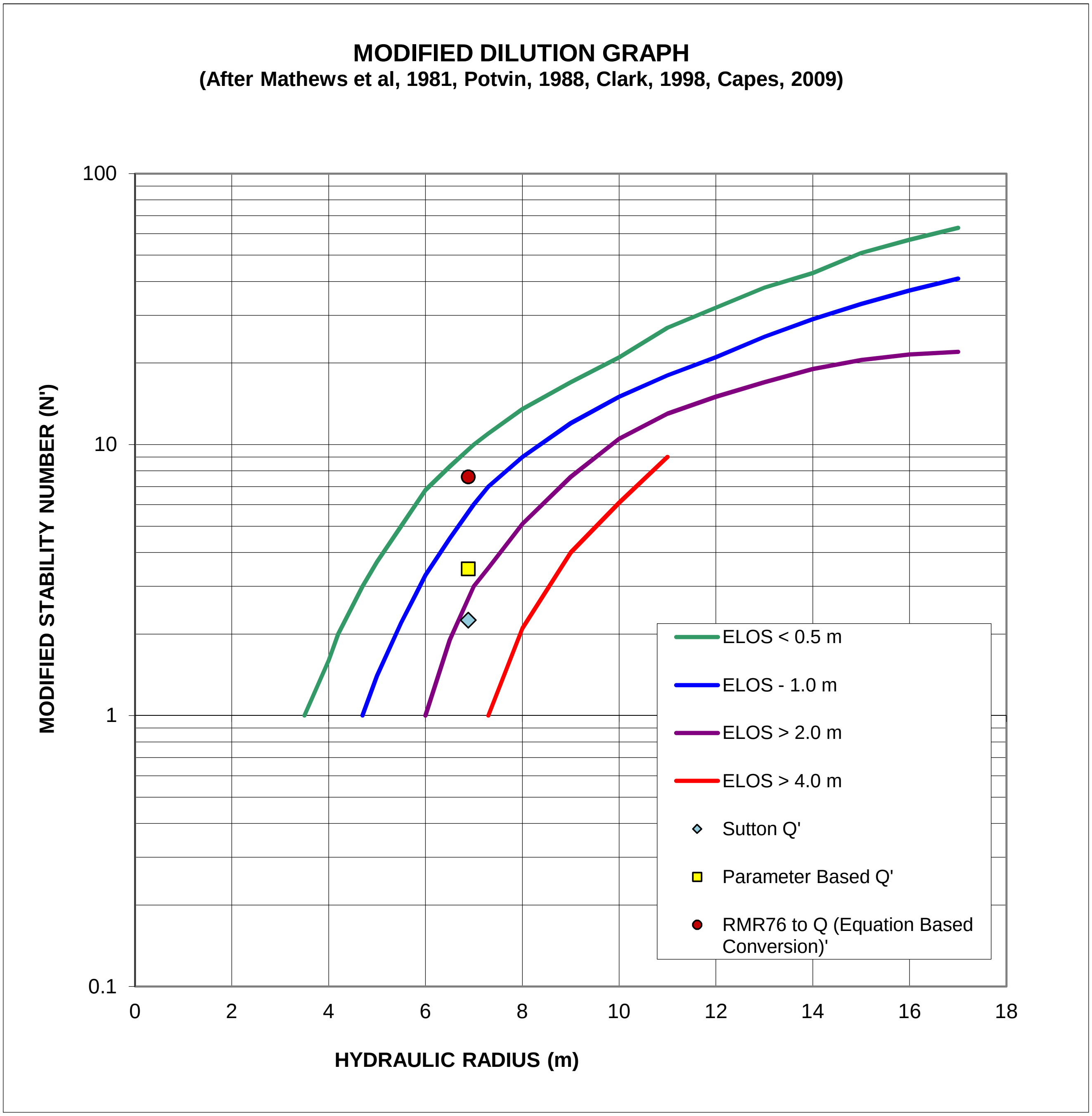
DDH's	RQD in HW:
2193	87

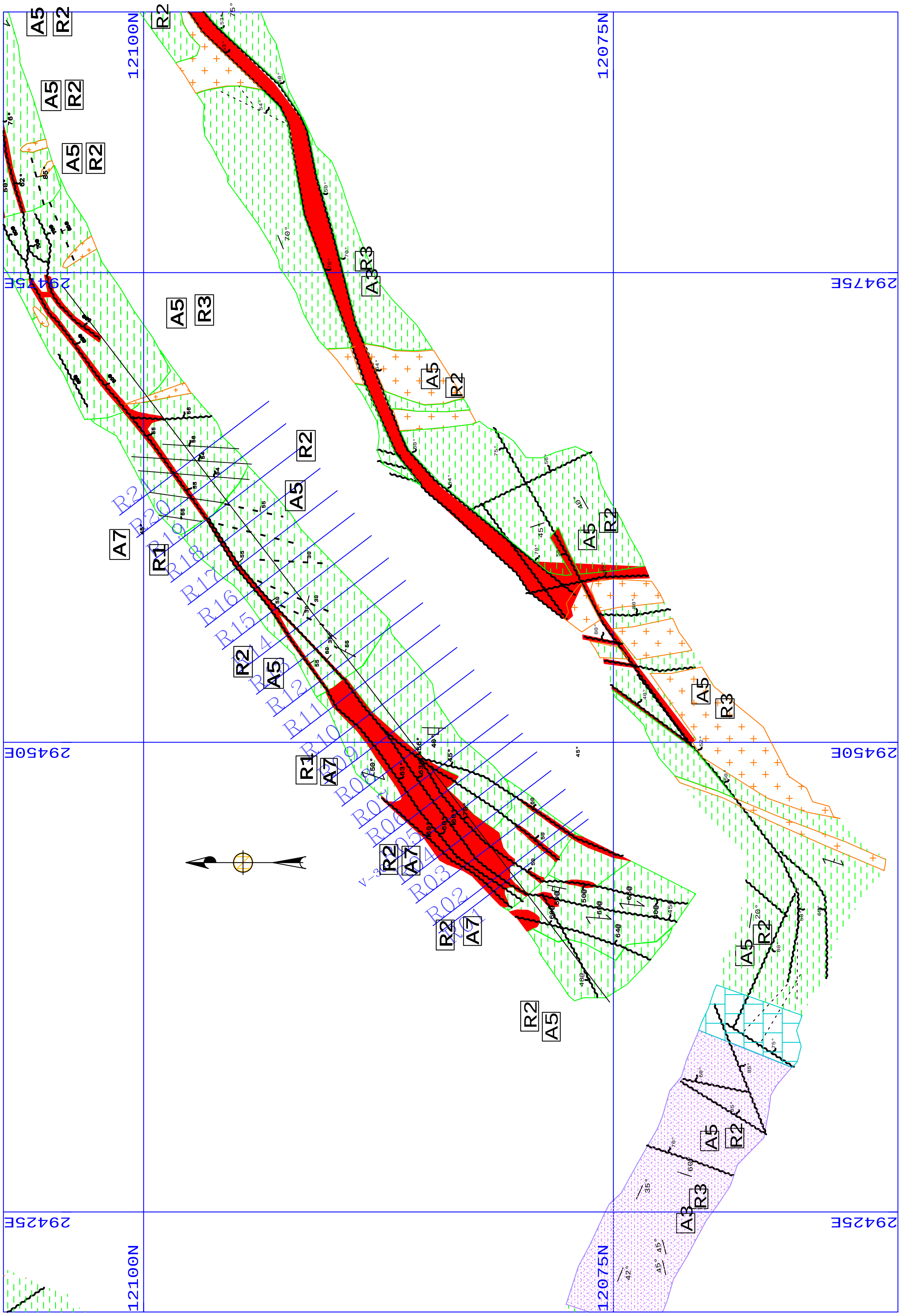
Q' Weighted	RQD	RQD O/C and U/C	Q'
8.7	75.8	70.2	8.1

A	B	C	N'
1	0.2	4.4	7.6

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	100-015	
Modified Stability Number (N')	Sutton N'	2.3
	Parameter Based N'	3.5
	RMR76 to Q (Equation Based Conversion)	7.6
Hydraulic Radius	Actual	6.9
Average HW Overbreak from the Optech:		**1.5** **m**




12100N
12075N
29475E
29450E
29425E
A5
R2
R3
A7
R1
A3

STOPE	100-045	
Strike Length	26	
HW Exposed Height	21.6	
Hydraulic Radius	5.9	
Average HW Dip	64.1	°
Percentage of HW that is:		
Gneissic	59	%
Pegmatoidal	41	%
Other	0	%

Sutton's Q' from A/R values

100-045 OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
GNEISS	A5/R2	8.9	75	667.5	7.5	66.75	1.5	13.35	6	53.4	
PEGMATOID	A5/R2	7.5	75	562.5	7.5	56.25	1.5	11.25	6	45	
	A3/R3	9.6	100	960	9	86.4	2.3	22.08	1	9.6	
TOTAL:		26		2190		209.4		46.68		108	
				84.2		8.1		1.8		4.2	4.5
UNDERCUT											
GNEISS	A1/R3	18	100	1800	9	162	1.5	27	1	18	
	A5/R3	4	100	400	9	36	1.5	6	2	8	
PEGMATOID	A1/R3	4	100	400	9	36	2	8	1	4	
TOTAL:		26		2600		234		41		30	
				100		9		1.6		1.2	15.2
									AVERAGE:		9.9

DDH's	RQD in HW:
2925	93
2922	66
2187	90

Q' Weighted	RQD	Jn	Jr	Ja
6.5	86.6	8.5	1.7	2.7

A	B	C	N'
1	0.2	4.6	6.0

Parameter Based Q' from A/R values

100-045											
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
GNEISS	A5/R2	8.9	75	667.5	9	80.1	1.5	13.35	3	26.7	
PEGMATOID	A5/R2	7.5	60	450	7.5	56.25	1.5	11.25	4	30	
	A3/R3	9.6	75	720	9	86.4	2.3	22.08	1	9.6	
TOTAL:		26		1837.5		222.75		46.68		66.3	
				70.7		8.6		1.8		2.6	5.8
UNDERCUT											
GNEISS	A1/R3	18	75	1350	9	162	1.5	27	1	18	
	A5/R3	4	75	300	9	36	1.5	6	2	8	
PEGMATOID	A1/R3	4	75	300	9	36	2	8	1	4	
TOTAL:		26		1950		234		41		30	
				75		9		1.6		1.2	11.4
									AVERAGE:		8.6

DDH's	RQD in HW:
2925	93
2922	66
2187	90

Q' Weighted	RQD	Jn	Jr	Ja
8.2	78.9	8.8	1.7	1.9

A	B	C	N'
1	0.2	4.6	7.6

100-045

RMR = 9 ln Q' + 44

OVERCUT		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
GNEISS	A5/R2	8.9	68	605.2	8.3	73.87
PEGMATOID	A5/R2	7.5	63	472.5	2.5	18.75
	A3/R3	9.6	62	595.2	6.2	59.52
TOTAL:		26		1672.9		152.14
				64.3		5.9
UNDERCUT						
GNEISS	A1/R3	18	63	1134	6.6	118.8
	A5/R3	4	73	292	14	56
PEGMATOID	A1/R3	4	73	292	15.2	60.8
TOTAL:		26		1718		235.6
				66.07692308		9.061538462

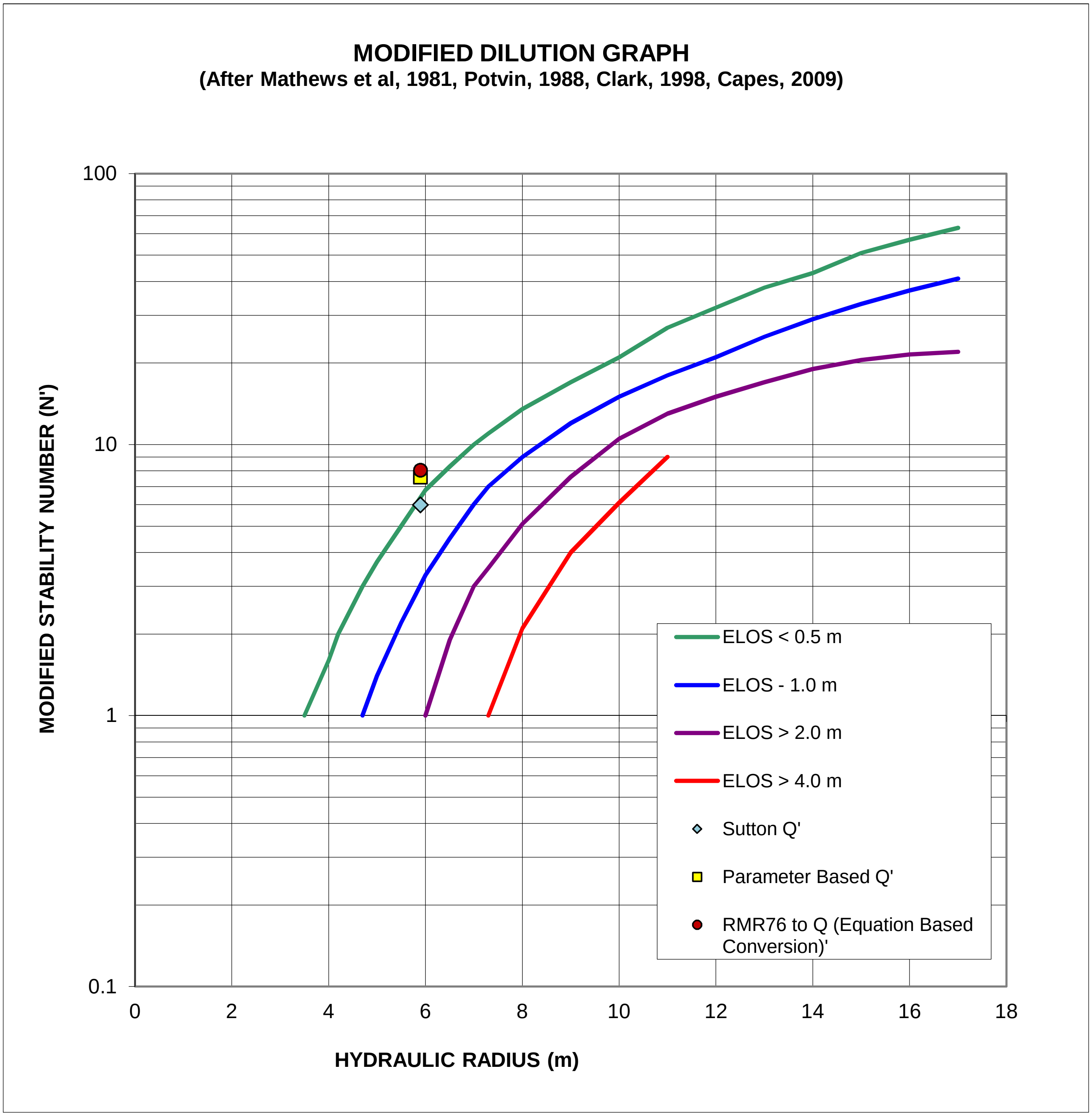
DDH's	RQD in HW:
2925	93
2922	66
2187	90

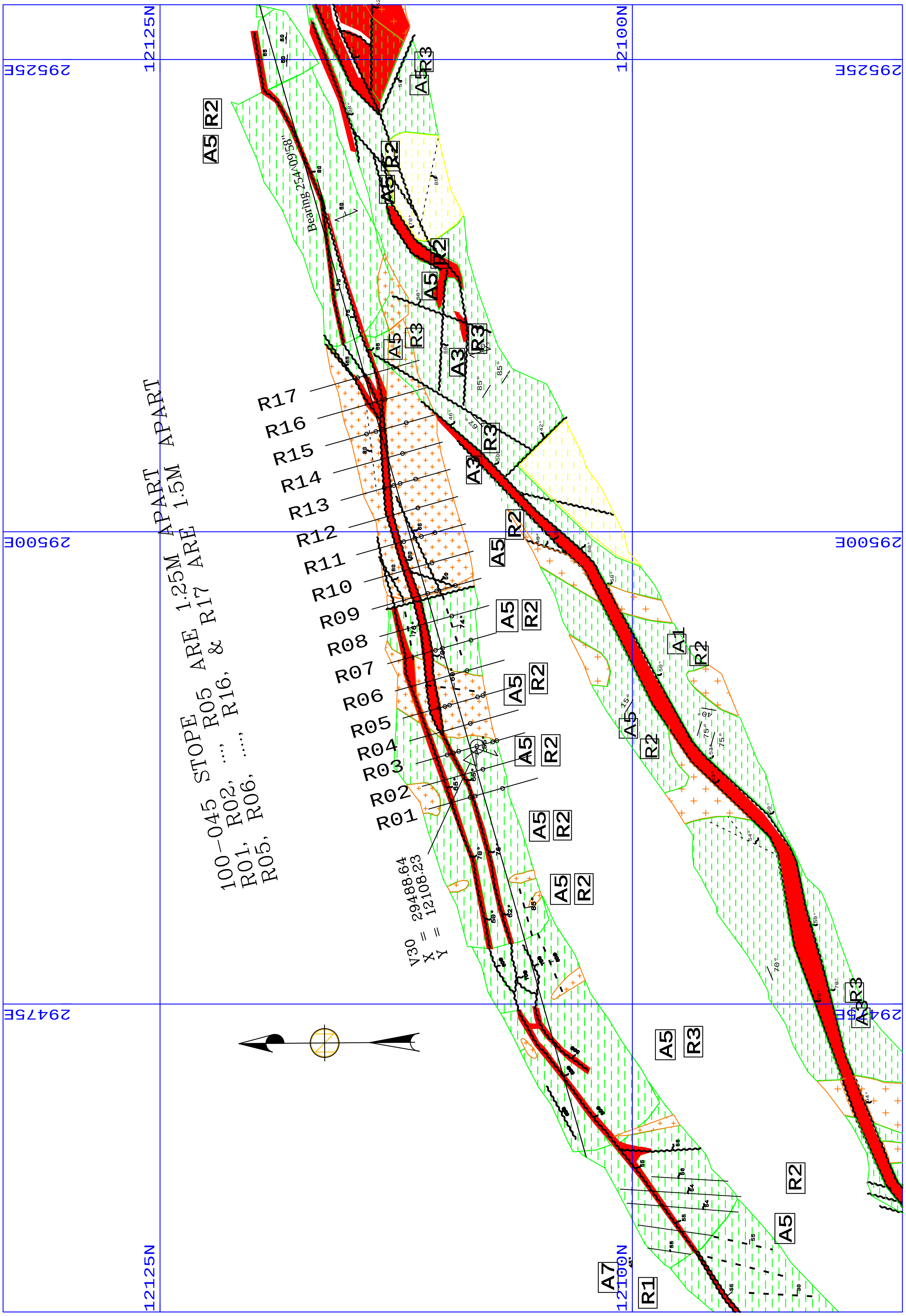
Q' Weighted	RQD	RQD O/C and U/C	Q'
8.7	75.9	65.2	7.5

A	B	C	N'
1	0.2	4.6	8.1

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	100-045	
Modified Stability Number (N')	Sutton N'	6.0
	Parameter Based N'	7.6
	RMR76 to Q (Equation Based Conversion)	8.1
Hydraulic Radius	Actual	5.9
Average HW Overbreak from the Optech:		**0.3** m




100-045 STOPE
R01, R02, ..., R05 ARE 1.25M APART
R05, R06, ..., R16, & R17 ARE 1.5M APART
12125N
12100N
29525E
29500E
29475E
Bearing 254°09'58''
V30
X = 29488.64
Y = 12108.23
R01
R02
R03
R04
R05
R06
R07
R08
R09
R10
R11
R12
R13
R14
R15
R16
R17
A5 R2
A5 R3
A3
A1 R2
A7
R1
R2

STOPE	100-075	
Strike Length	27	
HW Exposed Height	19.7	
Hydraulic Radius	5.7	
Average HW Dip	85.5	°
Percentage of HW that is:		
Gneissic	81	%
Pegmatoidal	19	%
Other	0	%

100-075			Sutton's Q' from A/R values								
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
BQFG	A3/R2	11	90	990	9	99	1.5	16.5	4	44	
	A3/R3	1	100	100	9	9	1.5	1.5	1.5	1.5	
	A5/R2	6	75	450	7.5	45	1.5	9	6	36	
PEGMATOID	A3/R2	3	90	270	9	27	1.5	4.5	4	12	
	A5/R1	6	50	300.0	4.0	24.0	0.8	4.5	8.0	48.0	
TOTAL:		27		2110		204		36		141.5	
				78.1		7.6		1.3		5.2	2.6
UNDERCUT											
BQFG	A5/R2	10	75	750	7.5	75	1.5	15	6	60	
	A7/R1	16	20	320	4	64	0.75	12	10	160	
PEGMATOID	A5/R2	1	75	75	7.5	7.5	1.5	1.5	6.0	6.0	
TOTAL:		27		1070		139		27		220	
				39.6		5.1		1.0		8.1	0.9
									AVERAGE:		1.8

DDH's	RQD in HW:
2910	90
2169	90
2912	79
2914.0	92.0

Q' Weighted	RQD	Jn	Jr	Ja
2.1	78.1	6.4	1.2	6.7

A	B	C	N'
1	0.2	7.81	3.3

100-075

Parameter Based Q' from A/R values

OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
BQFG	A3/R2	11	75	825	9	99	1.5	16.5	2	22	
	A3/R3	1	75	75	9	9	1.5	1.5	1.5	1.5	
	A5/R2	6	75	450	9	54	1.5	9	3	18	
PEGMATOID	A3/R2	3	75	225	9	27	1.5	4.5	3	9	
	A5/R1	6	40	240.0	4.0	24.0	0.75	4.5	8.0	48.0	
TOTAL:		27		1815		213		36		98.5	
				67.2		7.9		1.3		3.6	3.1
UNDERCUT											
BQFG	A5/R2	10	75	750	9	90	1.5	15	3	30	
	A7/R1	16	75	1200	9	144	0.75	12	4	64	
PEGMATOID	A5/R2	1	60	60	7.5	7.5	1.5	1.5	4.0	4.0	
TOTAL:		27		1950		234		27		94	
				72.2		8.7		1.0		3.5	2.4
									AVERAGE:		2.8

DDH's	RQD in HW:
2910	90
2169	90
2912	79
2914.0	92.0

Q' Weighted	RQD	Jn	Jr	Ja
3.2	81.7	8.3	1.2	3.6

A	B	C	N'
1	0.2	7.81	5.0

100-075

RMR = 9 ln Q' + 44

OVERCUT		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
BQFG	A3/R2	11	78	858	9.5	104.5
	A3/R3	1	77	77	15.7	15.7
	A5/R2	6	68	408	8.3	49.8
PEGMATOID	A3/R2	3	71	213	5.9	17.7
	A5/R1	6	25	150.0	0.3	1.8
TOTAL:		27		1706		189.5
				63.2		7.0

UNDERCUT						
BQFG	A5/R2	10	68	680	8.3	83
	A7/R1	16	69	1104	3.6	57.6
PEGMATOID	A5/R2	1	63	63	2.5	2.5
TOTAL:		27		1784		140.6
				66.1		5.2

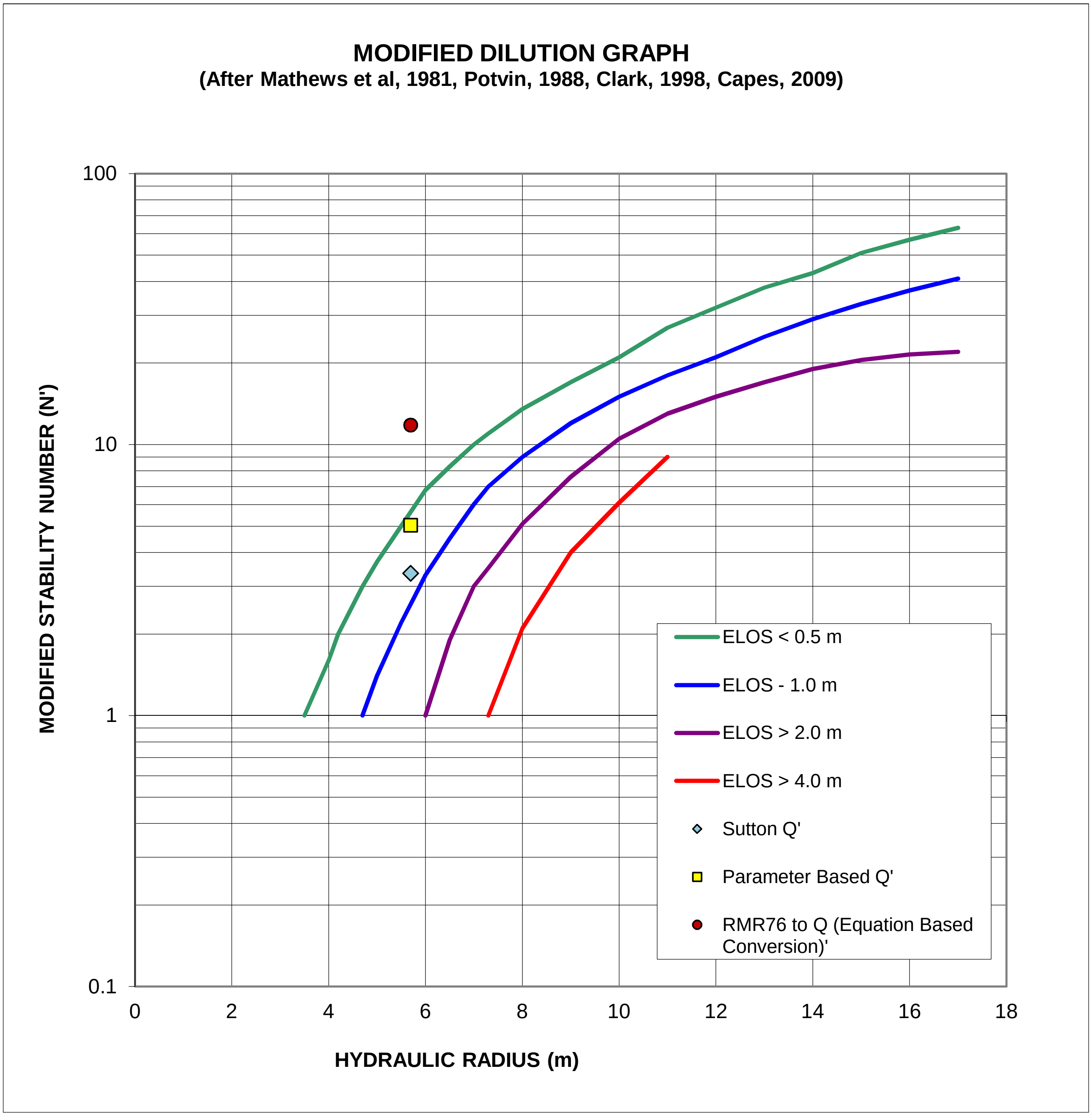
DDH's	RQD in HW:
2910	90
2169	90
2912	79
2914.0	92.0

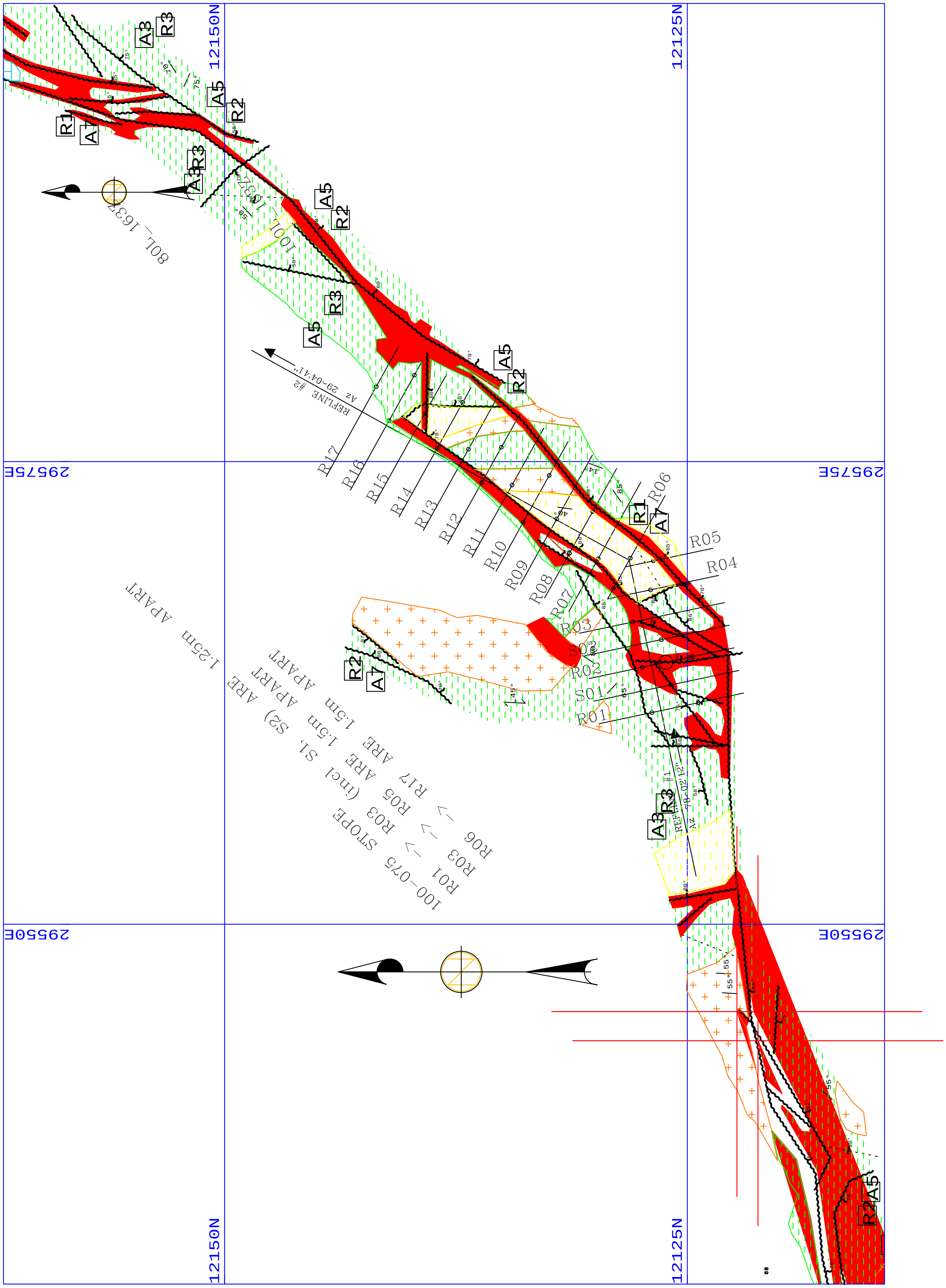
Q' Weighted	RQD	RQD O/C and U/C	Q'
7.6	80.0	64.6	6.1

A	B	C	N'
1	0.2	7.81	11.8

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	100-075	
Modified Stability Number (N')	Sutton N'	3.3
	Parameter Based N'	5.0
	RMR76 to Q (Equation Based Conversion)	11.8
Hydraulic Radius	Actual	5.7
Average HW Overbreak from the Optech:		**0.1** m




12150N
12125N
29575E
29550E
80L - 1632
100-075 STOPE
R01 <-> R03 (incl S1, S2) ARE 1.5m APART
R03 <-> R05 ARE 1.5m APART
R06 <-> R17 ARE 1.25m APART
REFLINE #2
REFLINE #1
R01
S01
R02
S02
R03
R04
R05
R06
R07
R08
R09
R10
R11
R12
R13
R14
R15
R16
R17
A3
A5
A7
R1
R2
R3

STOPE	100-085	
Strike Length	25	
HW Exposed Height	20.3	
Hydraulic Radius	5.6	
Average HW Dip	73.4	°
Percentage of HW that is:		
Gneissic	100	%
Pegmatoidal	0	%
Other	0	%

100-085			Sutton's Q' from A/R values								
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
BQFG	A5/R2	22	75	1650	7.5	165	1.5	33	6	132	
	A5/R3	3	100	300	9	27	1.5	4.5	2	6	
TOTAL:		25		1950		192		37.5		138	
				78		7.7		1.5		5.5	2.8
UNDERCUT											
BQFG	A5/R2	20	75	1500.0	7.5	150.0	1.5	30.0	6.0	120.0	
	A3/R3	5	100	500	9	45	1.5	7.5	1.5	7.5	
TOTAL:		25		2000		195		37.5		127.5	
				80		7.8		1.5		5.1	3.0
									AVERAGE:		2.9

DDH's	RQD in HW:
2500	33
2905	30
2907	83
2909	63

Q' Weighted	RQD	Jn	Jr	Ja
2.2	61.2	7.7	1.5	5.3

A	B	C	N'
1	0.2	6.3	2.8

100-085

Parameter Based Q' from A/R values

		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
OVERCUT											
BQFG	A5/R2	22	75	1650	9	198	1.5	33	3	66	
	A5/R3	3	75	225	9	27	1.5	4.5	2	6	
TOTAL:		25		1875		225		37.5		72	
				75		9		1.5		2.9	4.3
UNDERCUT											
BQFG	A5/R2	20	75	1500.0	9.0	180.0	1.5	30.0	3.0	60.0	
	A3/R3	5	75	375	9	45	1.5	7.5	1.5	7.5	
TOTAL:		25		1875		225		37.5		67.5	
				75		9		1.5		2.7	4.6
									AVERAGE:		4.5

DDH's	RQD in HW:
2500	33
2905	30
2907	83
2909	63

Q' Weighted	RQD	Jn	Jr	Ja
3.6	59.8	9	1.5	2.8

A	B	C	N'
1	0.2	6.3	4.5

100-085

RMR = 9 ln Q' + 44

		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
OVERCUT						
BQFG	A5/R2	22	68	1496	8.3	182.6
	A5/R3	3	73	219	14	42
TOTAL:		25		1715		224.6
				68.6		9.0
UNDERCUT						
BQFG	A5/R2	20	68	1360.0	8.3	166.0
	A3/R3	5	77	385	15.7	78.5
TOTAL:		25		1745		244.5
				69.8		9.8

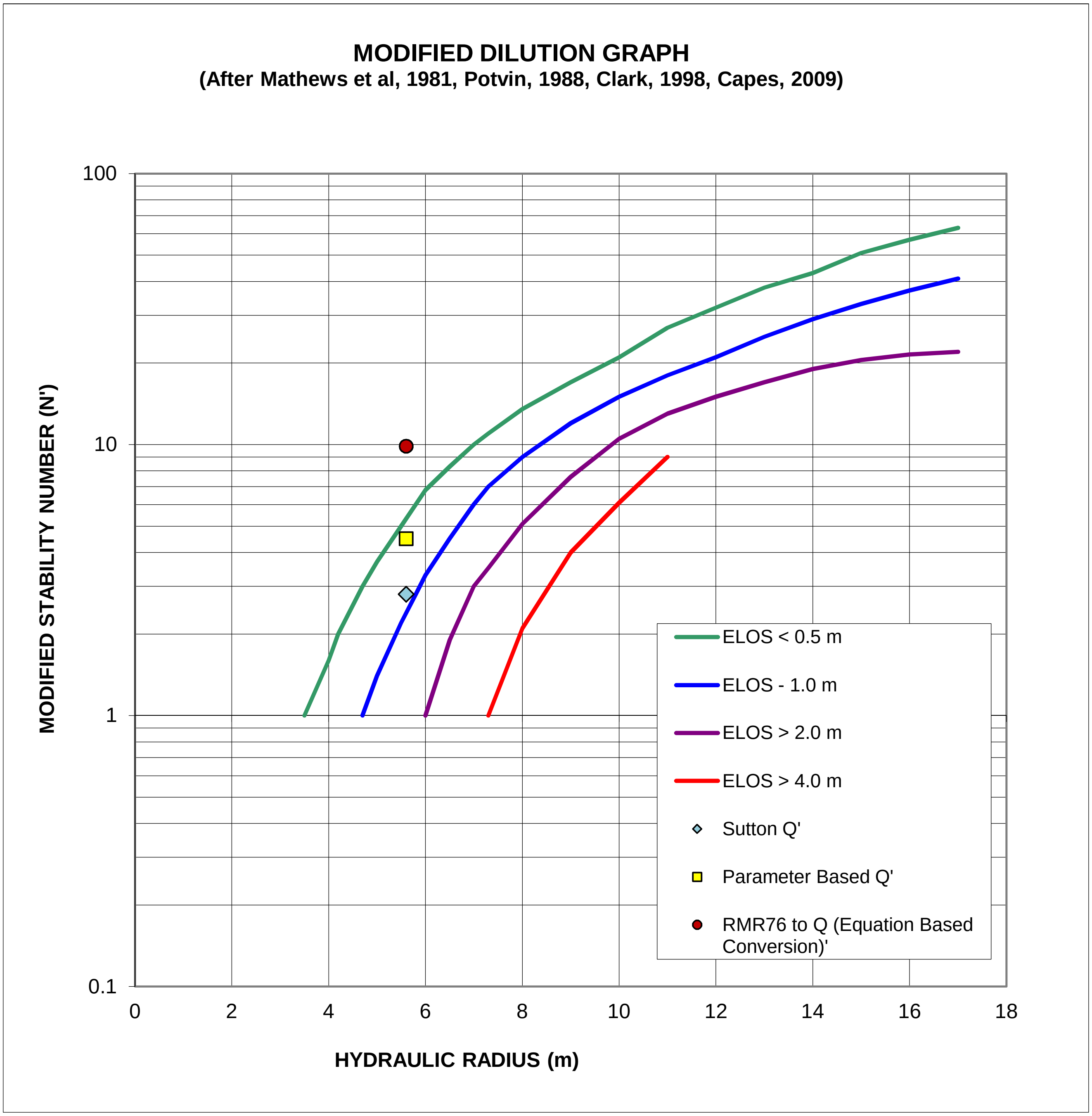
DDH's	RQD in HW:
2500	33
2905	30
2907	83
2909	63

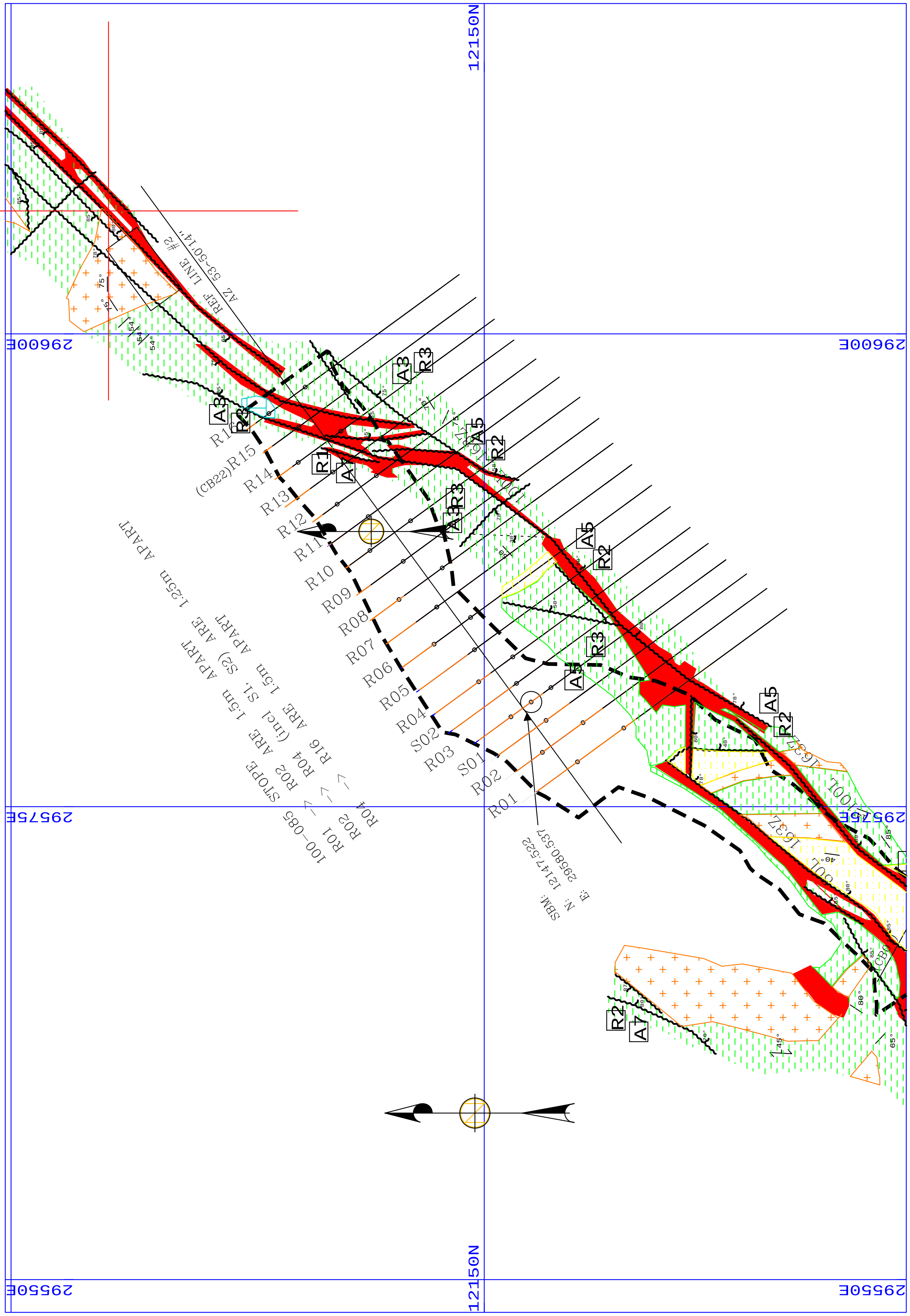
Q' Weighted	RQD	RQD O/C and U/C	Q'
7.8	57.9	69.2	9.4

A	B	C	N'
1	0.2	6.3	9.9

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	100-085	
Modified Stability Number (N')	Sutton N'	2.8
	Parameter Based N'	4.5
	RMR76 to Q (Equation Based Conversion)	9.9
Hydraulic Radius	Actual	5.6
Average HW Overbreak from the Optech:		**0.8** **m**





12150N
29600E
29575E
29550E
REF LINE #2
AZ 53-50'14"
R01
R02
S01
R03
S02
R04
R05
R06
R07
R08
R09
R10
R11
R12
R13
R14
(CB22)R15
R16
STOPE 100-085
R01 <-> R02 ARE 1.5m APART
R02 <-> R04 (incl S1, S2) ARE 1.5m APART
R04 <-> R16 ARE 1.25m APART
SDM: N: 12147.522
E: 29580.537
A3
R3
A5
R2
R1
A7

STOPE	102-1	
Strike Length	34	
HW Exposed Height	20.4	
Hydraulic Radius	6.4	
Average HW Dip	67.1	°
Percentage of HW that is:		
Gneissic	73	%
Pegmatoidal	0	%
Other	27	% (Calc Silicate, Feldspar Porphyry)

102-1			Sutton's Q' from A/R values								
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
BQFG	A7/R1	26.5	20	530	4	106	0.75	19.875	10	265	
	A7/R2	7.5	60	450	6	45	1.5	11.25	6	45	
TOTAL:		34		980		151		31.125		310	
				28.8		4.4		0.9		9.1	0.7
UNDERCUT											
FXPO	A5/R3	5	100	500.0	9.0	45.0	2.3	11.5	2.0	10.0	
	A5/R2	10	75	750	7.5	75	1.5	15	6	60	
CALC SIL	A5/R2	3.5	75	262.5	7.5	26.25	1.5	5.25	6	21	
BQFG	A5/R2	7	75	525	7.5	52.5	1.5	10.5	6.0	42.0	
	A7/R1	8.5	20	170	4	34	0.75	6.375	10	85	
TOTAL:		34		2207.5		232.75		48.6		218.0	
				64.9		6.8		1.4		6.4	2.1
				46.9		5.6		1.2		7.8	
									AVERAGE:		1.4

DDH's	RQD in HW:
1799	80

Q' Weighted	RQD	Jn	Jr	Ja
1.6	57.9	5.6	1.2	7.8

A	B	C	N'
1	0.2	5.4	1.7

102-1

Parameter Based Q' from A/R values

OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
BQFG	A7/R1	26.5	75	1987.5	9	238.5	0.75	19.875	4	106	
	A7/R2	7.5	75	562.5	9	67.5	1.5	11.25	4	30	
TOTAL:		34		2550		306		31.125		136	
				75		9		0.9		4.0	1.9
UNDERCUT											
FXPO	A5/R3	5	75	375.0	9.0	45.0	2.3	11.5	2.0	10.0	
	A5/R2	10	60	600	7.5	75	1.5	15	4	40	
CALC SIL	A5/R2	3.5	60	210	7.5	26.25	1.5	5.25	4	14	
BQFG	A5/R2	7	75	525	9	63	1.5	10.5	3	21	
	A7/R1	8.5	75	637.5	9	76.5	0.75	6.375	4	34	
TOTAL:		34		2347.5		285.75		48.6		119.0	
				69.0		8.4		1.4		3.5	3.4
				72.0		8.7		1.2		3.8	
									AVERAGE:		2.6

DDH's	RQD in HW:
1799	80

Q' Weighted	RQD	Jn	Jr	Ja
2.7	74.7	8.7	1.2	3.8

A	B	C	N'
1	0.2	5.4	2.9

102-1			RMR = 9 ln Q' + 44			
OVERCUT		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
BQFG	A7/R1	26.5	69	1828.5	3.6	95.4
	A7/R2	7.5	73	547.5	5	37.5
TOTAL:		34		2376		132.9
				69.9		3.9
UNDERCUT						
FXPO	A5/R3	5	79	395.0	18.0	90.0
	A5/R2	10	63	630	2.5	25
CALC SIL	A5/R2	3.5	63	220.5	2.5	8.75
BQFG	A5/R2	7	68	476	8.3	58.1
	A7/R1	8.5	69	586.5	3.6	30.6
TOTAL:		34		2308		212.45
				67.9		6.2

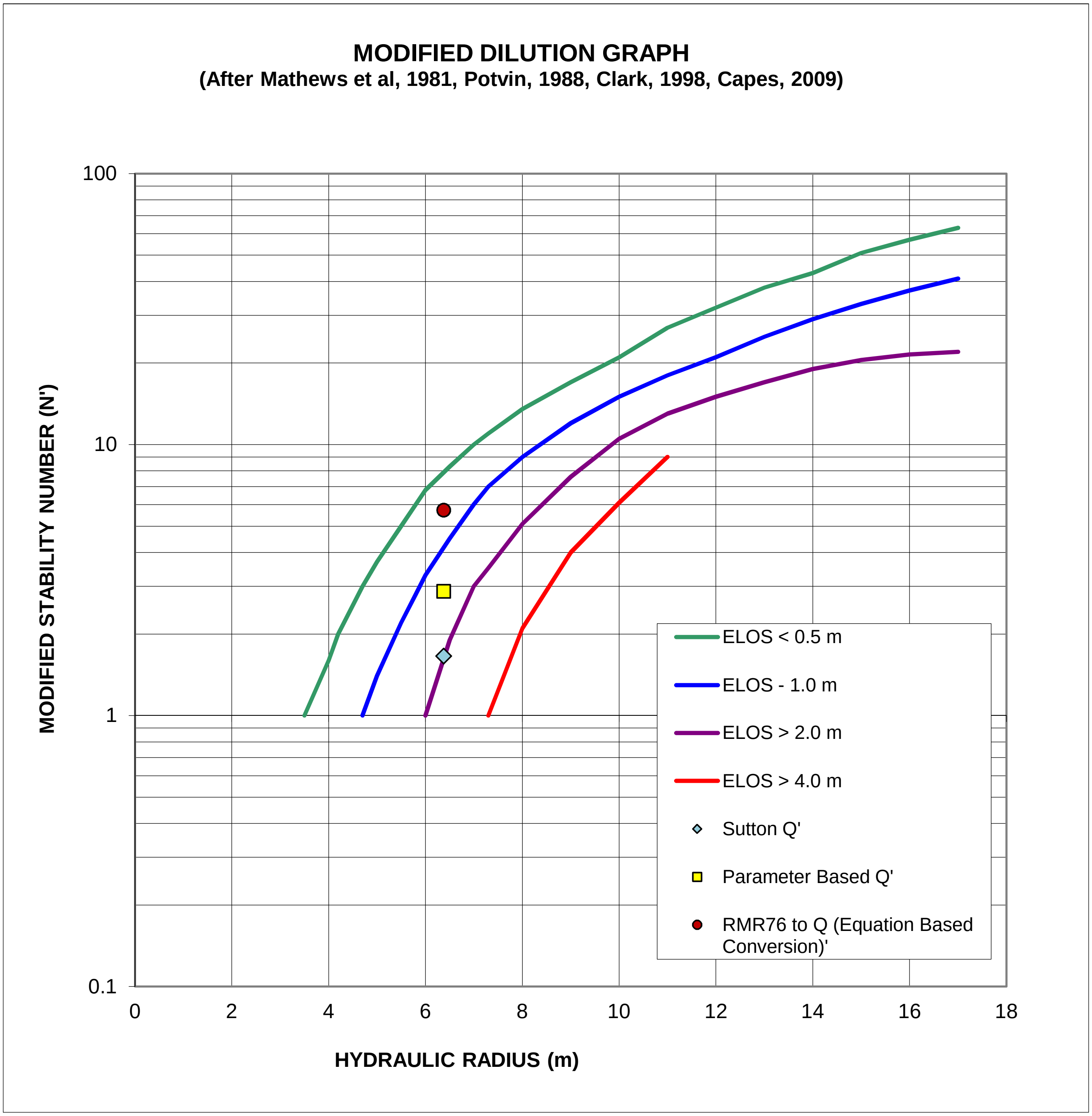
DDH's	RQD in HW:
1799	80

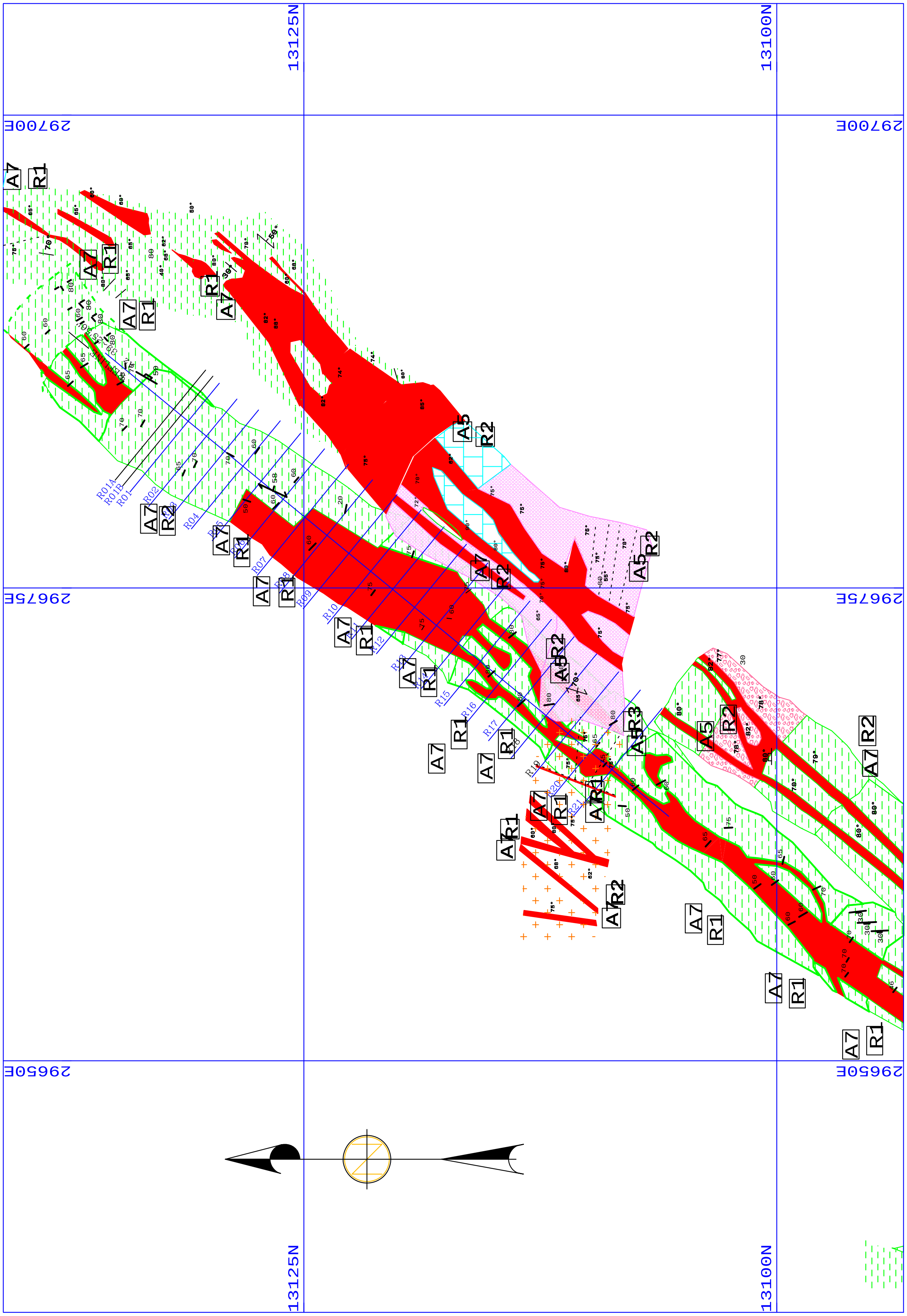
Q' Weighted	RQD	RQD O/C and U/C	Q'
5.4	72.6	68.9	5.1

A	B	C	N'
1	0.2	5.4	5.7

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	102-1	
Modified Stability Number (N')	Sutton N'	1.7
	Parameter Based N'	2.9
	RMR76 to Q (Equation Based Conversion)	5.7
Hydraulic Radius	Actual	6.4
Average HW Overbreak from the Optech:		**0.7** m





13125N
13100N
29700E
29675E
29650E
A7
R1
R2
A5
R3

STOPE	102-2	
Strike Length	30	
HW Exposed Height	21.2	
Hydraulic Radius	6.2	
Average HW Dip	71.2	°
Percentage of HW that is:		
Gneissic	92	%
Pegmatoidal	0	%
Other	8	% (Feldspar Porphyry)

Sutton's Q' from A/R values

102-2											
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
BQFG	A7/R1	24	20	480	4	96	0.75	18	10	240	
	A5/R2	5	75	375	7.5	37.5	1.5	7.5	6	30	
	A7/R2	1	60	60	6	6	1.5	1.5	6	6	
TOTAL:		30		915.0		139.5		27.0		276.0	
				30.5		4.7		0.9		9.2	0.6
UNDERCUT											
BQFG	A5/R2	18	75	1350	7.5	135	1.5	27	6	108	
	A7/R2	7	60	420	6	42	1.5	10.5	6	42	
FXPO	A7/R2	5	60	300	6	30.0	1.5	7.5	6.0	30.0	
TOTAL:		30		2070		207		45		180	
				69		6.9		1.5		6.0	2.5

DDH's	RQD in HW:
1785	47
1835	80
1817	90

Q' Weighted	RQD	Jn	Jr	Ja
1.7	63.3	5.8	1.2	7.6

A	B	C	N'
1	0.2	6.1	2.1

102-2

Parameter Based Q' from A/R values

OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
BQFG	A7/R1	24	75	1800	9	216	0.75	18	4	96	
	A5/R2	5	75	375	9	45	1.5	7.5	3	15	
	A7/R2	1	75	75	9	9	1.5	1.5	4	4	
TOTAL:		30		2250		270		27.0		115.0	
				75.0		9.0		0.9		3.8	2.0
UNDERCUT											
BQFG	A5/R2	18	75	1350	9	162	1.5	27	3	54	
	A7/R2	7	75	525	9	63	1.5	10.5	4	28	
FXPO	A7/R2	5	50	250	6	30	1.5	7.5	6	30	
TOTAL:		30		2125		255		45		112	
				70.83333333		8.5		1.5		3.7	3.3

DDH's	RQD in HW:
1785	47
1835	80
1817	90

Q' Weighted	RQD	Jn	Jr	Ja
2.6	72.6	8.8	1.2	3.8

A	B	C	N'
1	0.2	6.1	3.2

102-2

$RMR = 9 \ln Q' + 44$

OVERCUT		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
BQFG	A7/R1	24	69	1656	3.6	86.4
	A5/R2	5	68	340	8.3	41.5
	A7/R2	1	73	73	5	5
TOTAL:		30		2069.0		132.9
				69.0		4.4

UNDERCUT						
BQFG	A5/R2	18	68	1224	8.3	149.4
	A7/R2	7	73	511	5	35
FXPO	A7/R2	5	73	365	4.7	23.5
TOTAL:		30		2100		207.9
				70		6.93

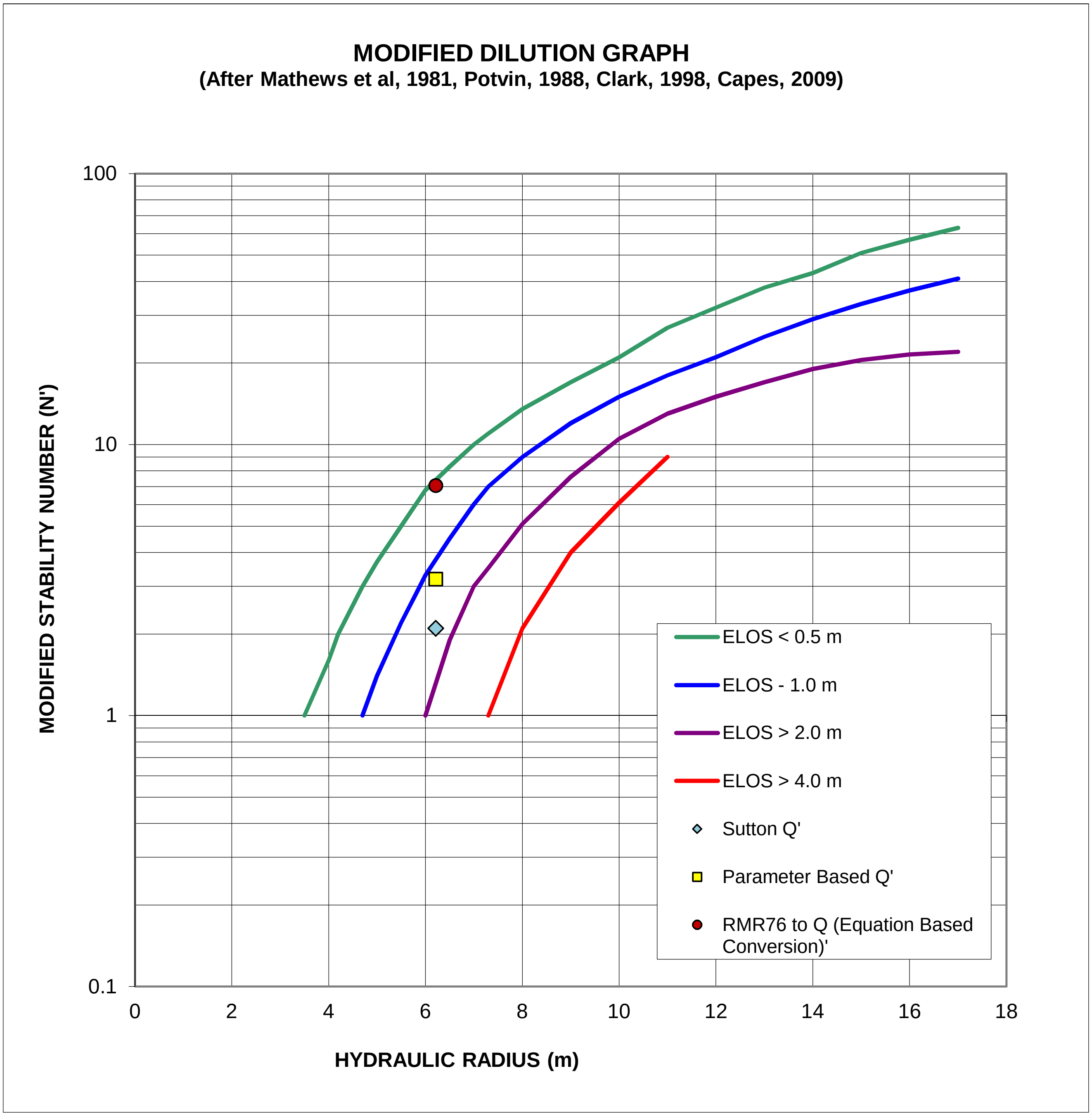
DDH's	RQD in HW:
1785	47
1835	80
1817	90

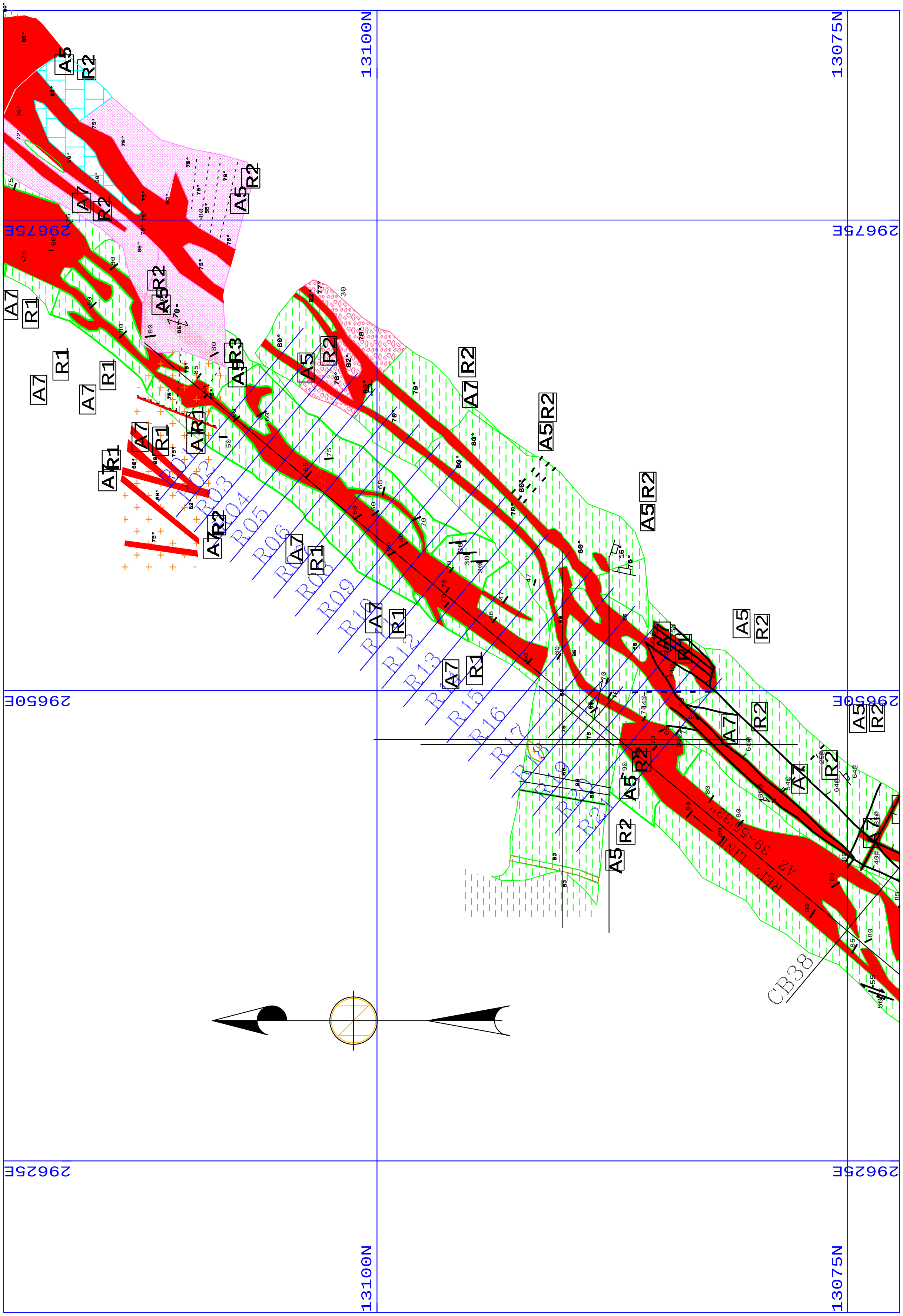
Q' Weighted	RQD	RQD O/C and U/C	Q'
5.8	71.2	69.5	5.7

A	B	C	N'
1	0.2	6.1	7.1

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	102-2	
Modified Stability Number (N')	Sutton N'	2.1
	Parameter Based N'	3.2
	RMR76 to Q (Equation Based Conversion)	7.1
Hydraulic Radius	Actual	6.2
Average HW Overbreak from the Optech:		**0.7** m




13100N
13075N
29675E
29650E
29625E
A5
R2
A7
R1
R3
CB38

STOPE	122-1	
Strike Length	29.5	
HW Exposed Height	20.1	
Hydraulic Radius	6.0	
Average HW Dip	87.7	°
Percentage of HW that is:		
Gneissic	81	%
Pegmatoidal	0	%
Other	19	% (Calc-Silicate, Feldspar Porphyry)

Sutton's Q' from A/R values

122-1											
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
FXPO	A5/R2	5.5	75	412.5	7.5	41.25	1.5	8.25	6	33	
CALC-SIL	A5/R2	6	75	450	7.5	45	1.5	9	6	36	
BQFG	A7/R1	18	20	360	4	72	0.75	13.5	10	180	
TOTAL:		29.5		1222.5		158.3		30.8		249.0	
				41.4		5.4		1.0		8.4	1.0
UNDERCUT											
BQFG	A5/R2	14.5	75	1087.5	7.5	108.75	1.5	21.75	6	87	
	A7/R1	15	20	300	4	60	0.75	11.25	10	150	
TOTAL:		29.5		1387.5		168.8		33.0		237.0	
				47.0		5.7		1.1		8.0	1.1

DDH's	RQD in HW:
1801	53
1903	77

Q' Weighted	RQD	Jn	Jr	Ja
1.3	54.6	5.5	1.1	8.2

A	B	C	N'
1	0.2	7.8	2.0

122-1

Parameter Based Q' from A/R values

OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
FXPO	A5/R2	5.5	60	330	7.5	41.25	1.5	8.25	4	22	
CALC-SIL	A5/R2	6	60	360	7.5	45	1.5	9	4	24	
BQFG	A7/R1	18	75	1350	9	162	0.75	13.5	4	72	
TOTAL:		29.5		2040		248.25		30.8		118.0	
				69.2		8.4		1.0		4.0	2.1
UNDERCUT											
BQFG	A5/R2	14.5	75	1087.5	9	130.5	1.5	21.75	3	43.5	
	A7/R1	15	75	1125	9	135	0.75	11.25	4	60	
TOTAL:		29.5		2212.5		265.5		33		103.5	
				75		9		1.1		3.5	2.7

DDH's	RQD in HW:
1801	53
1903	77

Q' Weighted	RQD	Jn	Jr	Ja
2.3	68.5	8.7	1.1	3.8

A	B	C	N'
1	0.2	7.76	3.5

122-1

RMR = 9 ln Q' + 44

OVERCUT		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
FXPO	A5/R2	5.5	63	346.5	2.5	13.75
CALC-SIL	A5/R2	6	63	378	2.5	15
BQFG	A7/R1	18	69	1242	3.6	64.8
TOTAL:		29.5		1966.5		93.6
				66.7		3.2
UNDERCUT						
BQFG	A5/R2	14.5	68	986	8.3	120.35
	A7/R1	15	69	1035	3.6	54
TOTAL:		29.5		2021		174.4
				68.5		5.9

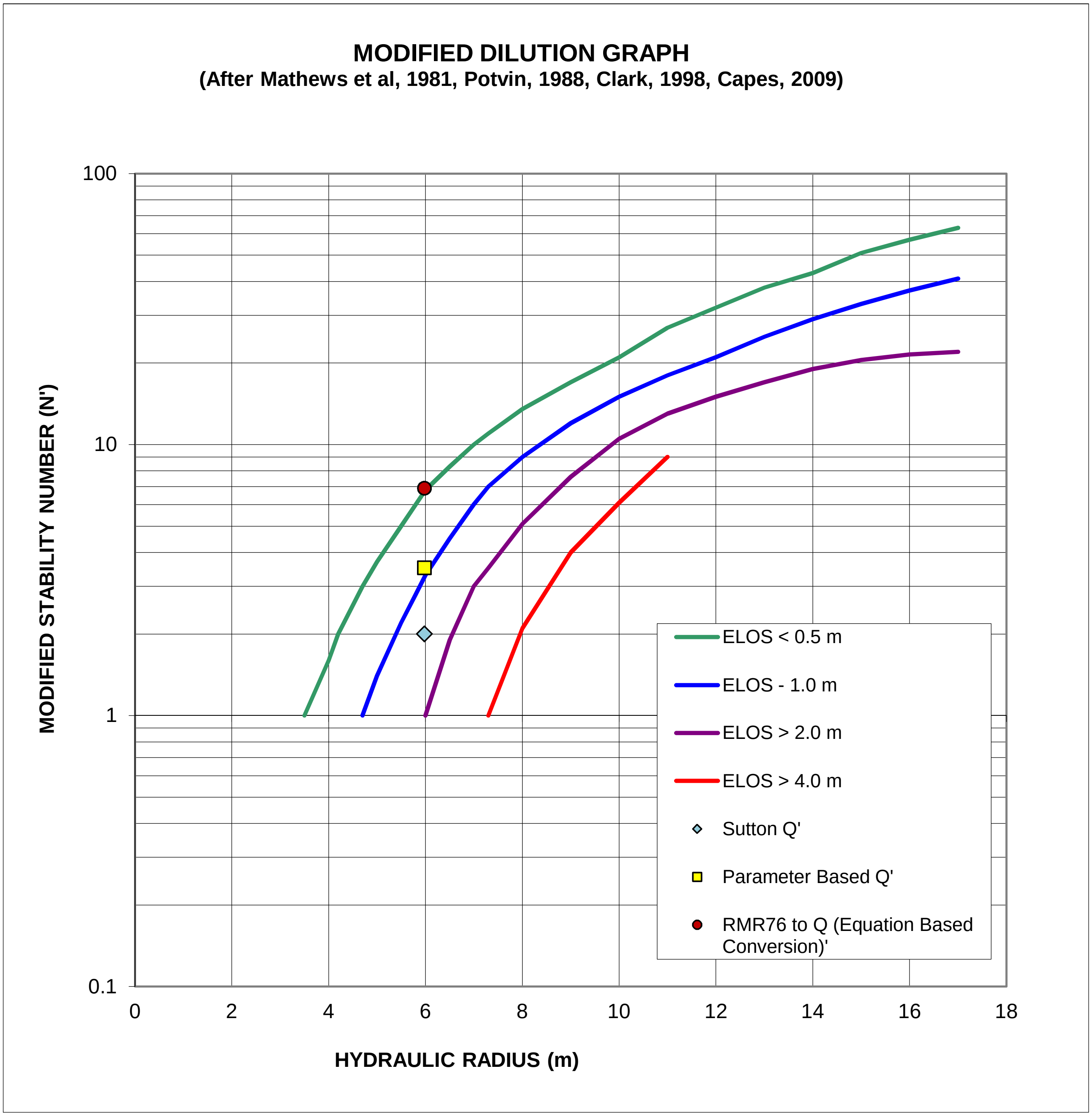
DDH's	RQD in HW:
1801	53
1903	77

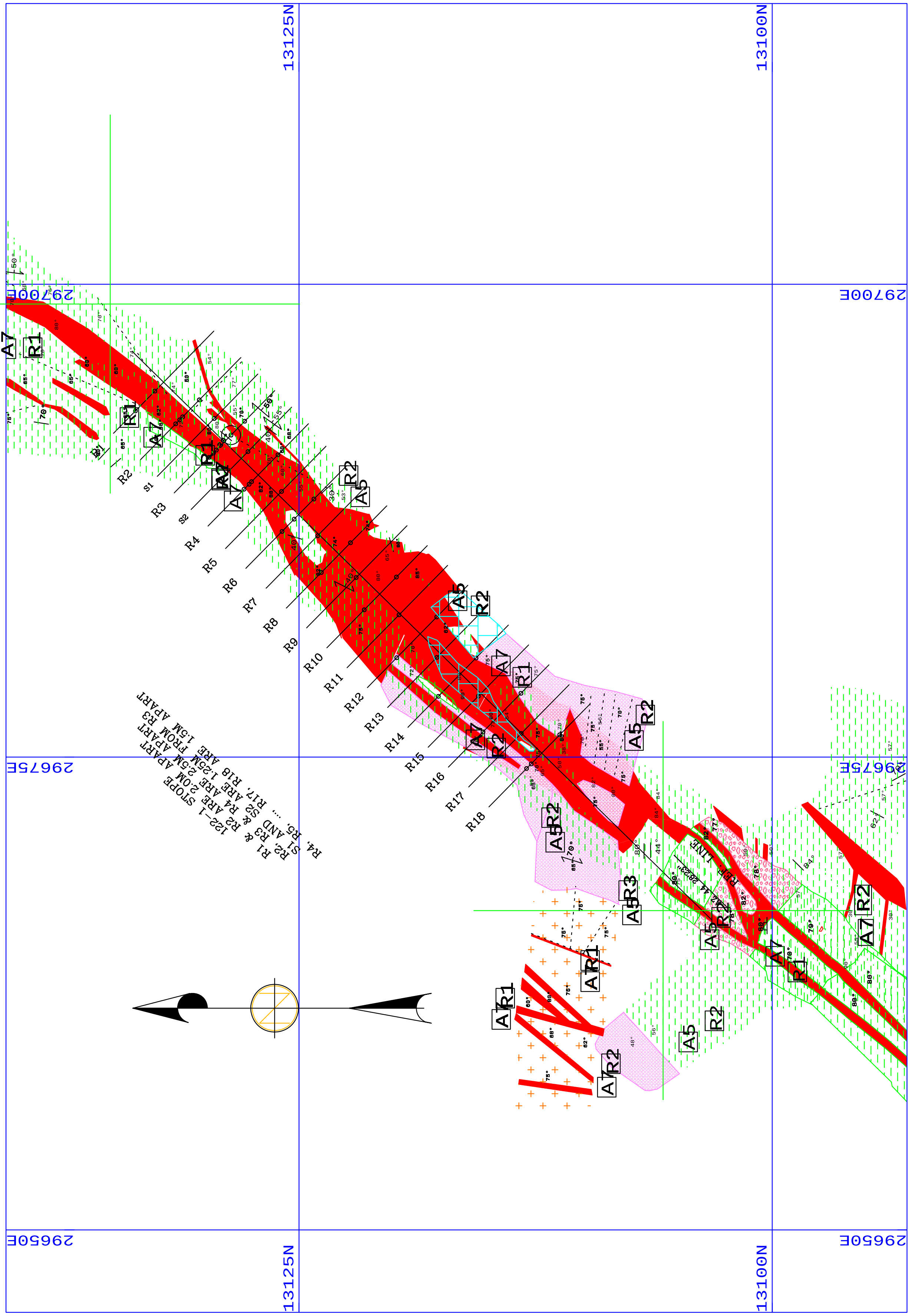
Q' Weighted	RQD	RQD O/C and U/C	Q'
4.5	66.3	67.6	4.5

A	B	C	N'
1	0.2	7.8	6.9

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	122-1	
Modified Stability Number (N')	Sutton N'	2.0
	Parameter Based N'	3.5
	RMR76 to Q (Equation Based Conversion)	6.9
Hydraulic Radius	Actual	6.0
Average HW Overbreak from the Optech:		**1.7** m




13125N
13100N
29700E
29675E
29650E

STOPE	122-3	
Strike Length	23	
HW Exposed Height	20.6	
Hydraulic Radius	5.4	
Average HW Dip	80.0	°
Percentage of HW that is:		
Gneissic	100	%
Pegmatoidal	0	%
Other	0	%

122-3			Sutton's Q' from A/R values								
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
BQFG	A5/R2	23	75	1725	7.5	172.5	1.5	34.5	6	138	
TOTAL:		23		1725		172.5		34.5		138	
				75		7.5		1.5		6	2.5
UNDERCUT											
BQFG	A5/R2	23	75	1725	7.5	172.5	1.5	34.5	6	138	
TOTAL:		23		1725.0		172.5		34.5		138.0	
				75		7.5		1.5		6	2.5
				75		7.5		1.5		6	
									AVERAGE:		2.5

DDH's	RQD in HW:
1819	63
1837	97

Q' Weighted	RQD	Jn	Jr	Ja
2.6	77.5	7.5	1.5	6.0

A	B	C	N'
1.0	0.2	7.3	3.8

122-3

Parameter Based Q' from A/R values

		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
OVERCUT											
BQFG	A5/R2	23	75	1725	9	207	1.5	34.5	3	69	
TOTAL:		23		1725		207		34.5		69	
				75		9		1.5		3	4.2
UNDERCUT											
BQFG	A5/R2	23	75	1725	9	207	1.5	34.5	3	69	
TOTAL:		23		1725.0		207.0		34.5		69.0	
				75		9		1.5		3	4.2
				75		9		1.5		3	
									AVERAGE:		4.2

DDH's	RQD in HW:
1819	63
1837	97

Q' Weighted	RQD	Jn	Jr	Ja
4.3	77.5	9	1.5	3.0

A	B	C	N'
1.0	0.2	7.3	6.3

122-3			RMR = 9 ln Q' + 44			
OVERCUT		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
BQFG	A5/R2	23	68	1564	8.3	190.9
TOTAL:		23		1564		190.9
				68		8.3
UNDERCUT						
BQFG	A5/R2	23	68	1564	8.3	190.9
TOTAL:		23		1564.0		190.9
				68		8.3

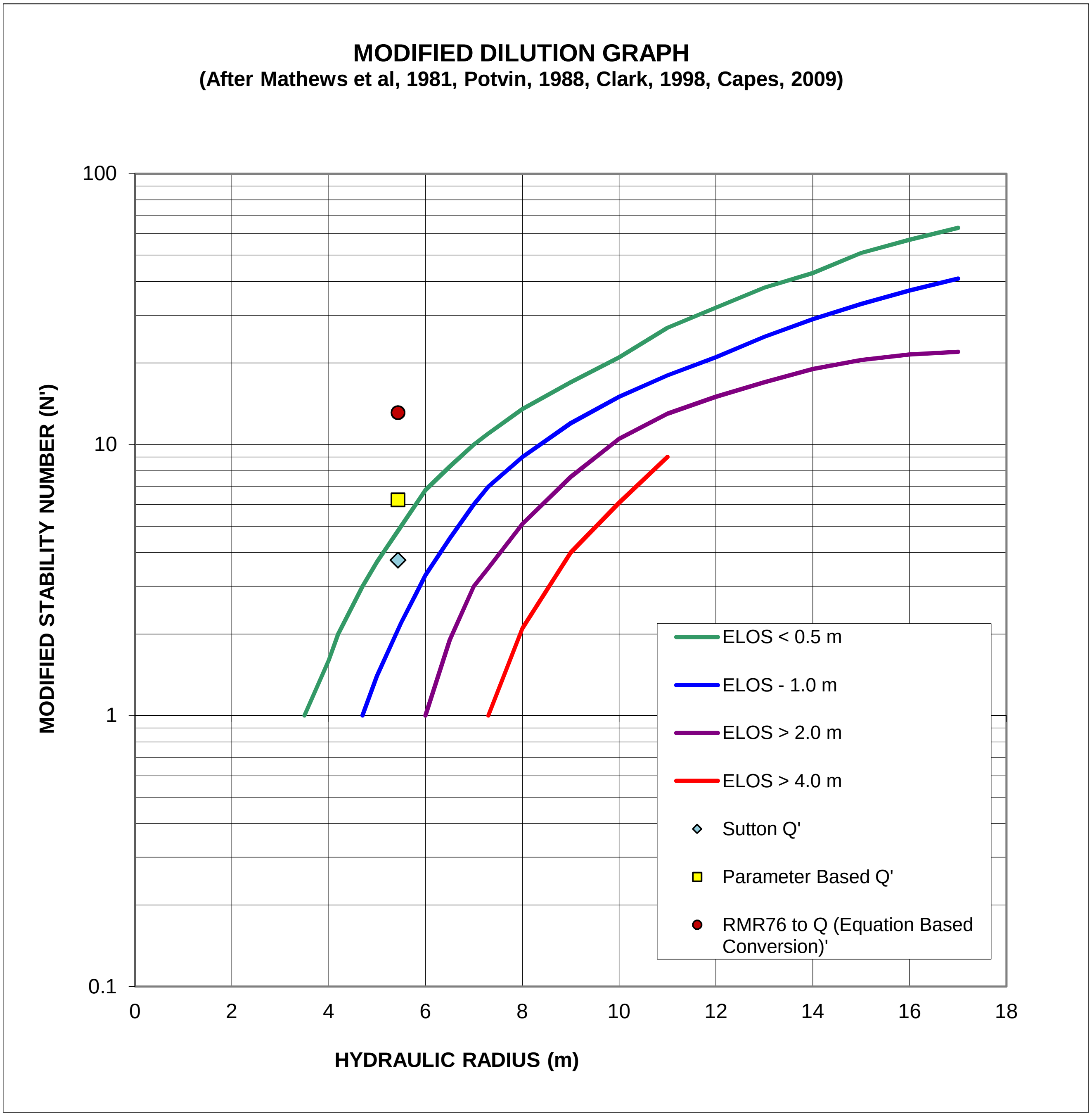
DDH's	RQD in HW:
1819	63
1837	97

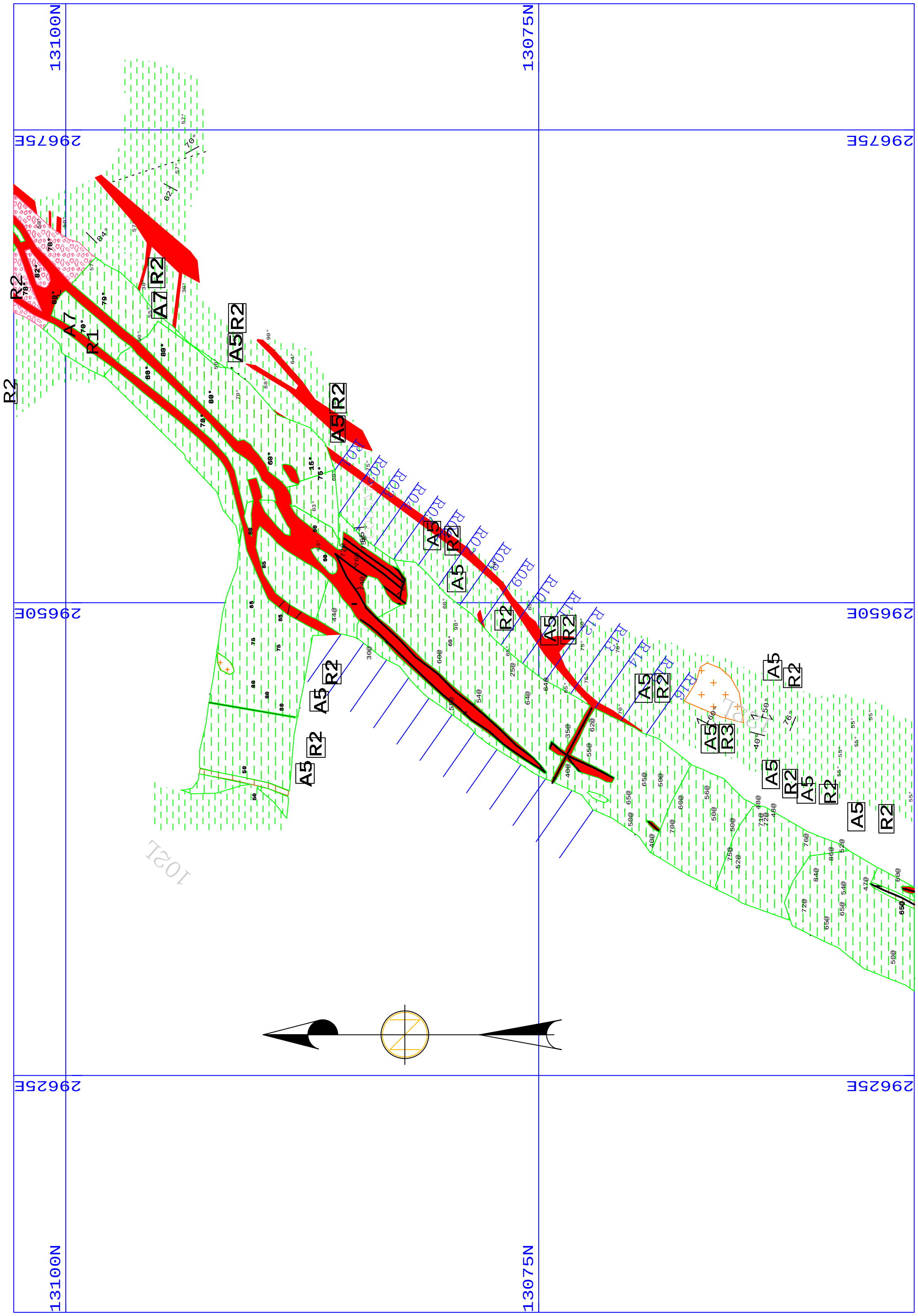
Q' Weighted	RQD	RQD O/C and U/C	Q'
9.032352941	74	68	8.3

A	B	C	N'
1.0	0.2	7.3	13.1

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	122-3	
Modified Stability Number (N')	Sutton N'	3.8
	Parameter Based N'	6.3
	RMR76 to Q (Equation Based Conversion)	13.1
Hydraulic Radius	Actual	5.4
Average HW Overbreak from the Optech:		**1.0** **m**




13100N
13075N
29675E
29650E
29625E
A5
R2
A7
R1
R3

STOPE	125-675	
Strike Length	18.5	
HW Exposed Height	27.8	
Hydraulic Radius	5.6	
Average HW Dip	38.6	°
Percentage of HW that is:		
Gneissic	22	%
Pegmatoidal	78	%
Other	0	%

125-675			Sutton's Q' from A/R values								
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
Pegmatoid	A7/R2	10.5	60	630	6	63	1.5	15.75	6	63	
BQFG	A7/R2	4.5	60	270	6	27	1.5	6.75	6	27	
	A5/R2	3.5	75	262.5	7.5	26.25	1.5	5.25	6	21	
TOTAL:		18.5		1162.5		116.3		27.8		111.0	
				62.8		6.3		1.5		6.0	2.5
UNDERCUT											
Pegmatoid	A5/R2	14	75	1050	7.5	105	1.5	21	6	84	
	A7/R1	4.5	20	90	4	18	0.75	3.375	10	45	
TOTAL:		18.5		1140		123.0		24.4		129.0	
				61.6		6.6		1.3		7.0	1.8

DDH's	RQD in HW:
3485	66
3486	100
3492	90

Q' Weighted	RQD	Jn	Jr	Ja
2.6	76.1	6.5	1.4	6.5

A	B	C	N'
1	0.2	3.3	1.7

125-675

Parameter Based Q' from A/R values

		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
OVERCUT											
Pegmatoid	A7/R2	10.5	50	525	6	63	1.5	15.75	6	63	
BQFG	A7/R2	4.5	75	337.5	9	40.5	1.5	6.75	4	18	
	A5/R2	3.5	75	262.5	9	31.5	1.5	5.25	3	10.5	
TOTAL:		18.5		1125		135		27.8		91.5	
				60.8		7.3		1.5		4.9	2.5
UNDERCUT											
Pegmatoid	A5/R2	14	60	840	7.5	105	1.5	21	4	56	
	A7/R1	4.5	25	112.5	4	18	0.75	3.375	10	45	
TOTAL:		18.5		952.5		123		24.375		101	
				51.5		6.6		1.3		5.5	1.9

DDH's	RQD in HW:
3485	66
3486	100
3492	90

Q' Weighted	RQD	Jn	Jr	Ja
2.9	73.7	7.0	1.4	5.2

A	B	C	N'
1	0.2	3.3	1.9

125-675			RMR = 9 ln Q' + 44			
OVERCUT		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
Pegmatoid	A7/R2	10.5	73	766.5	4.7	49.35
BQFG	A7/R2	4.5	73	328.5	5	22.5
	A5/R2	3.5	68	238	8.3	29.05
TOTAL:		18.5		1333.0		100.9
				72.1		5.5
UNDERCUT						
Pegmatoid	A5/R2	14	63	882	2.5	35
	A7/R1	4.5	38	171	0.3	1.35
TOTAL:		18.5		1053		36.4
				56.9		2.0

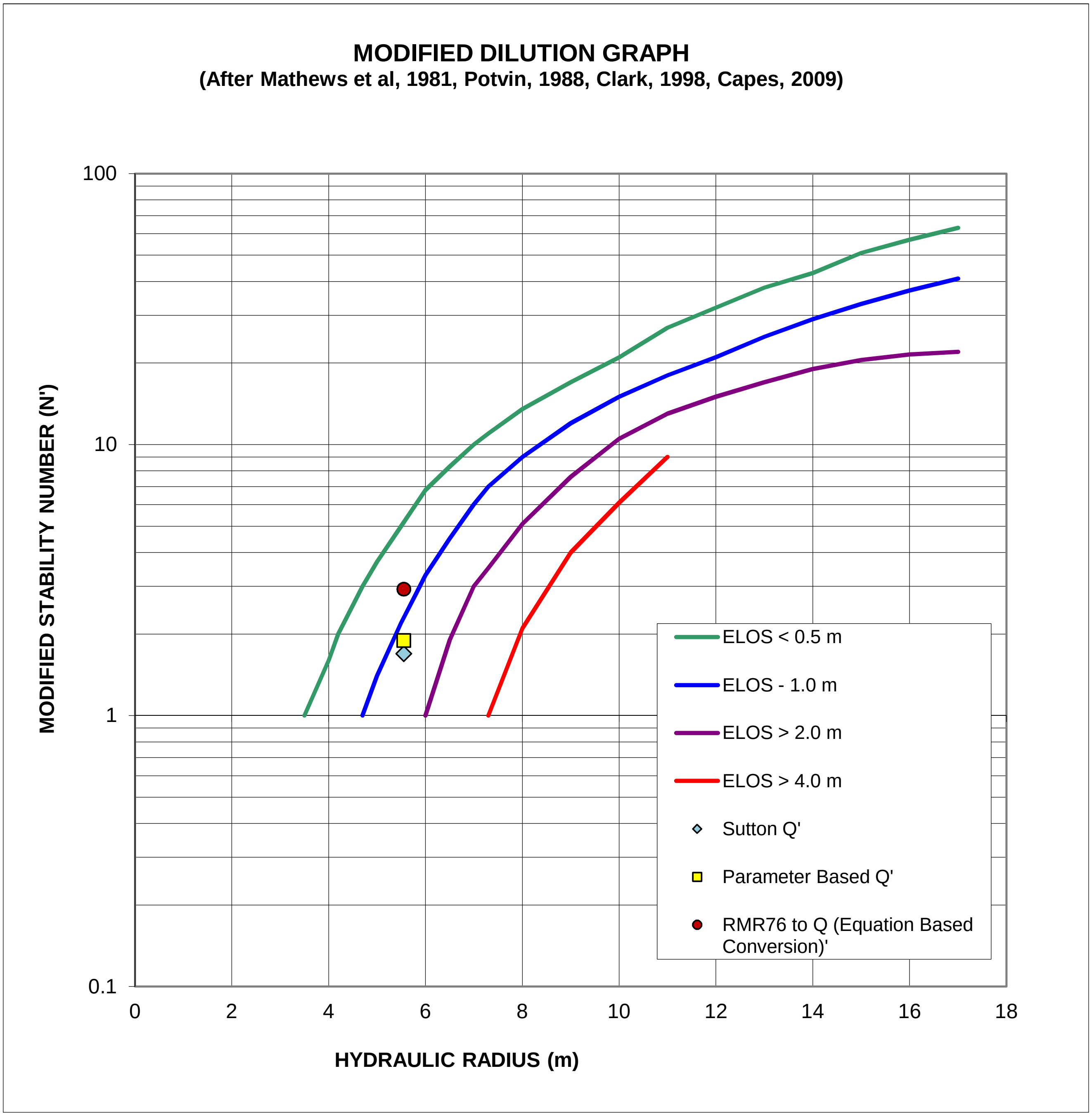
DDH's	RQD in HW:
3485	66
3486	100
3492	90

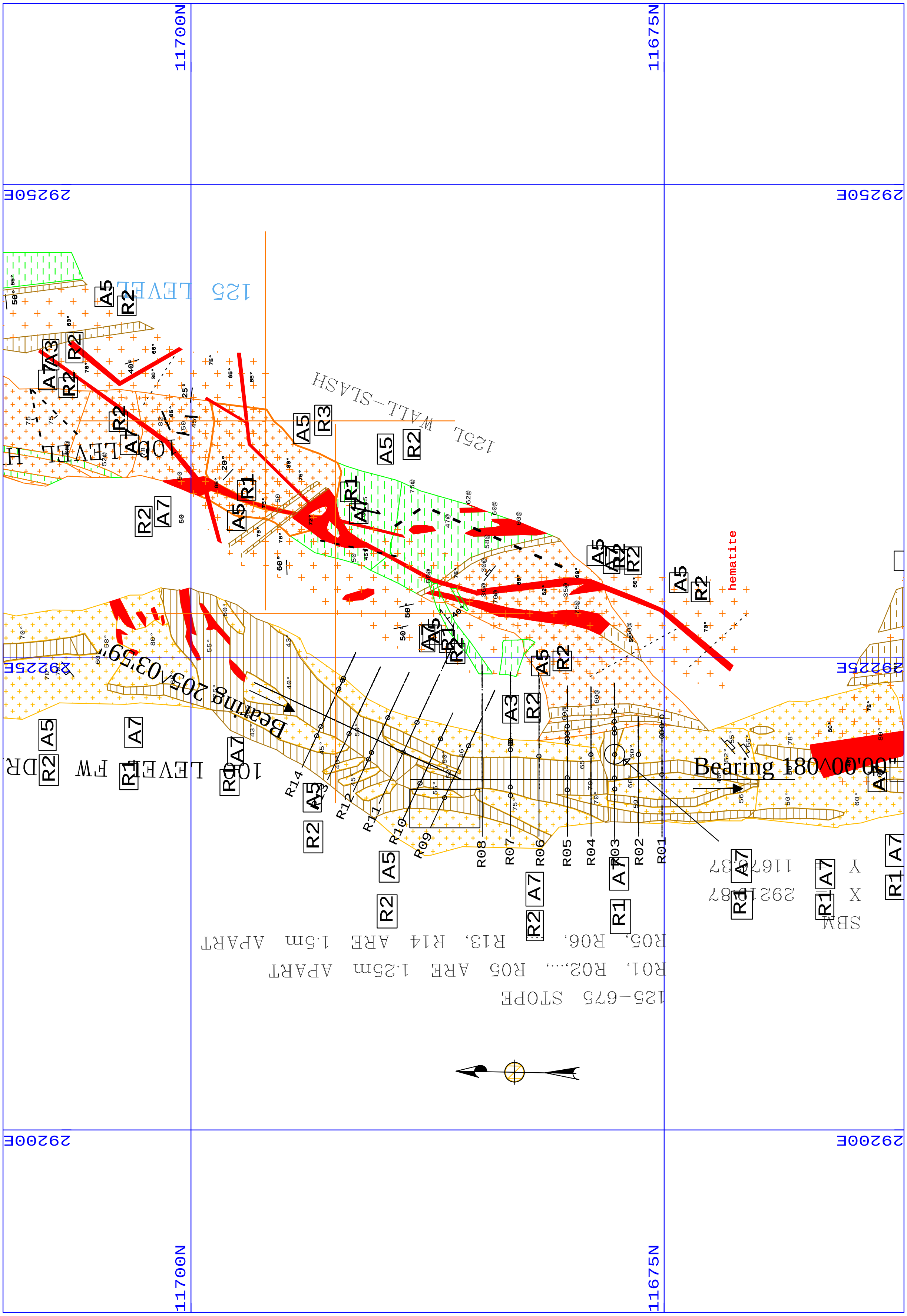
Q' Weighted	RQD	RQD O/C and U/C	Q'
4.4	77.0	64.5	3.7

A	B	C	N'
1	0.2	3.3	2.9

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	125-675	
Modified Stability Number (N')	Sutton N'	1.7
	Parameter Based N'	1.9
	RMR76 to Q (Equation Based Conversion)	2.9
Hydraulic Radius	Actual	5.6
Average HW Overbreak from the Optech:		**2.7** m




11700N
11675N
29250E
29225E
29200E
125 LEVEL
125L WALL-SLASH
100 LEVEL FW DR
hematite
Bearing 180^00'00"
Bearing 205^03'59"
125-675 STOPE
R01, R02..... R05 ARE 1.25m APART
R05, R06, ... R13, R14 ARE 1.5m APART
SBM

STOPE	125-695	
Strike Length	23	
HW Exposed Height	28.3	
Hydraulic Radius	6.3	
Average HW Dip	38.4	°
Percentage of HW that is:		
Gneissic	16	%
Pegmatoidal	84	%
Other	0	%

125-695

Sutton's Q' from A/R values

OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
Pegmatoid	A3/R2	9.5	90	855	9	85.5	1.5	14.25	4	38	
	A5/R3	9	100	900	9	81	2.3	20.7	2	18	
BQFG	A5/R2	4.5	75	337.5	7.5	33.75	1.5	6.75	6	27	
TOTAL:		23		2092.5		200.3		41.7		83.0	
				91.0		8.7		1.8		3.6	5.2
UNDERCUT											
BQFG	A5/R2	3	75	225	7.5	22.5	1.5	4.5	6	18	
Pegmatoid	A5/R2	9.5	75	712.5	7.5	71.25	1.5	14.25	6	57	
	A7/R1	10.5	20	210	4	42.0	0.8	7.9	10.0	105.0	
TOTAL:		23		1147.5		135.8		26.6		180.0	
				49.9		5.9		1.2		7.8	1.3

DDH's	RQD in HW:
3472	70
3473	93
3481	60

Q' Weighted	RQD	Jn	Jr	Ja
2.6	72.8	7.3	1.5	5.7

A	B	C	N'
1	0.2	3.3	1.7

Parameter Based Q' from A/R values

125-695											
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
Pegmatoid	A3/R2	9.5	75	712.5	9	85.5	1.5	14.25	3	28.5	
	A5/R3	9	75	675	9	81	2.3	20.7	2	18	
BQFG	A5/R2	4.5	75	337.5	9	40.5	1.5	6.75	3	13.5	
TOTAL:		23		1725		207		41.7		60.0	
				75.0		9.0		1.8		2.6	5.8
UNDERCUT											
BQFG	A5/R2	3	75	225	9	27	1.5	4.5	3	9	
Pegmatoid	A5/R2	9.5	60	570	7.5	71.25	1.5	14.25	4	38	
	A7/R1	10.5	25	262.5	4	42	0.75	7.875	10	105	
TOTAL:		23		1057.5		140.3		26.6		152.0	
				46.0		6.1		1.2		6.6	1.3

DDH's	RQD in HW:
3472	70
3473	93
3481	60

Q' Weighted	RQD	Jn	Jr	Ja
2.9	68.8	7.5	1.5	4.6

A	B	C	N'
1	0.2	3.3	1.9

RMR = 9 ln Q' + 44

125-695		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
OVERCUT						
Pegmatoid	A3/R2	9.5	71	674.5	5.9	56.05
	A5/R3	9	79	711	18	162
BQFG	A5/R2	4.5	68	306	8.3	37.35
TOTAL:		23		1691.5		255.4
				73.5		11.1
UNDERCUT						
BQFG	A5/R2	3	68	204	8.3	24.9
Pegmatoid	A5/R2	9.5	63	598.5	2.5	23.75
	A7/R1	10.5	38	399	0.3	3.2
TOTAL:		23		1201.5		51.8
				52.2		2.3

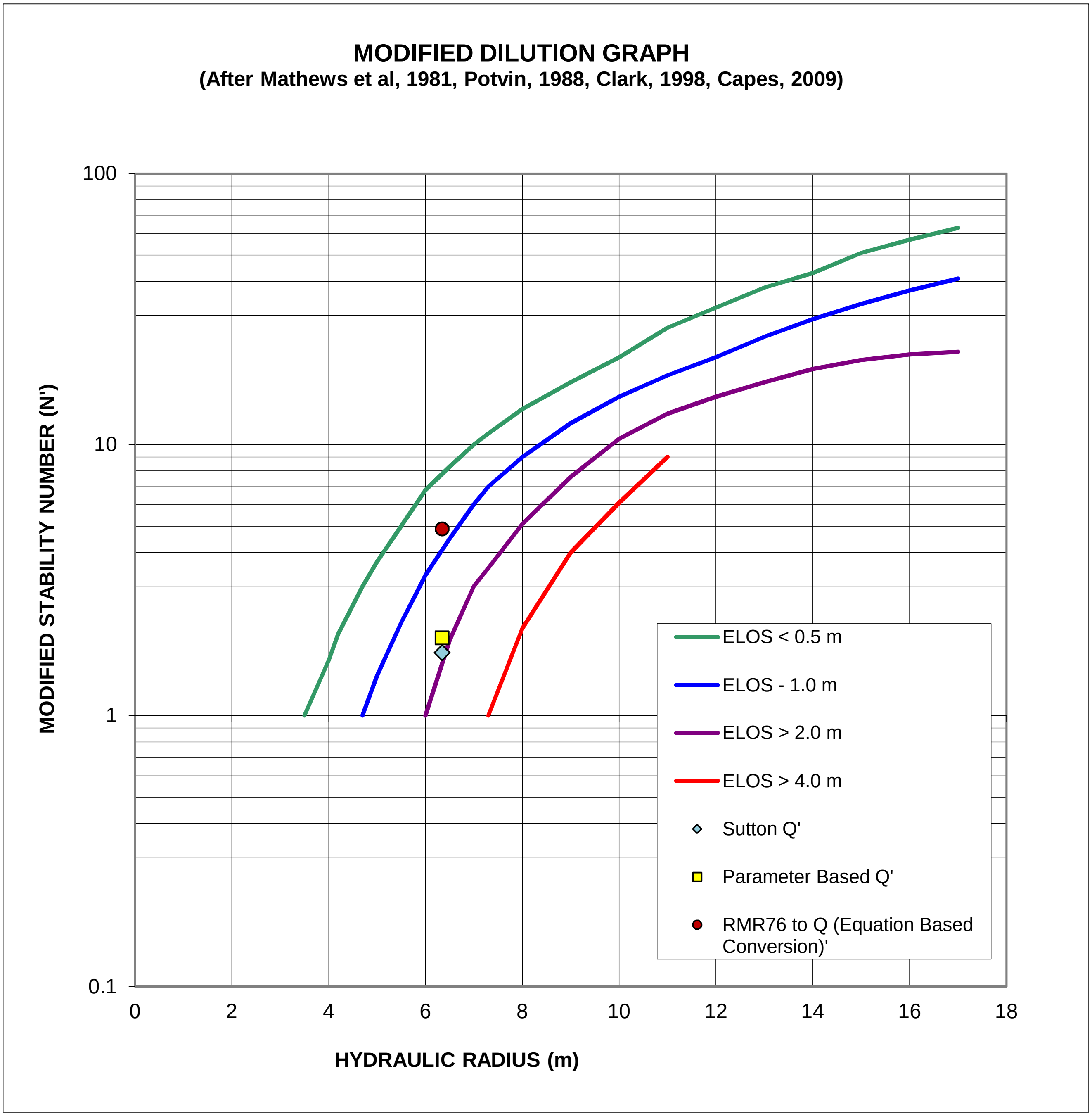
DDH's	RQD in HW:
3472	70
3473	93
3481	60

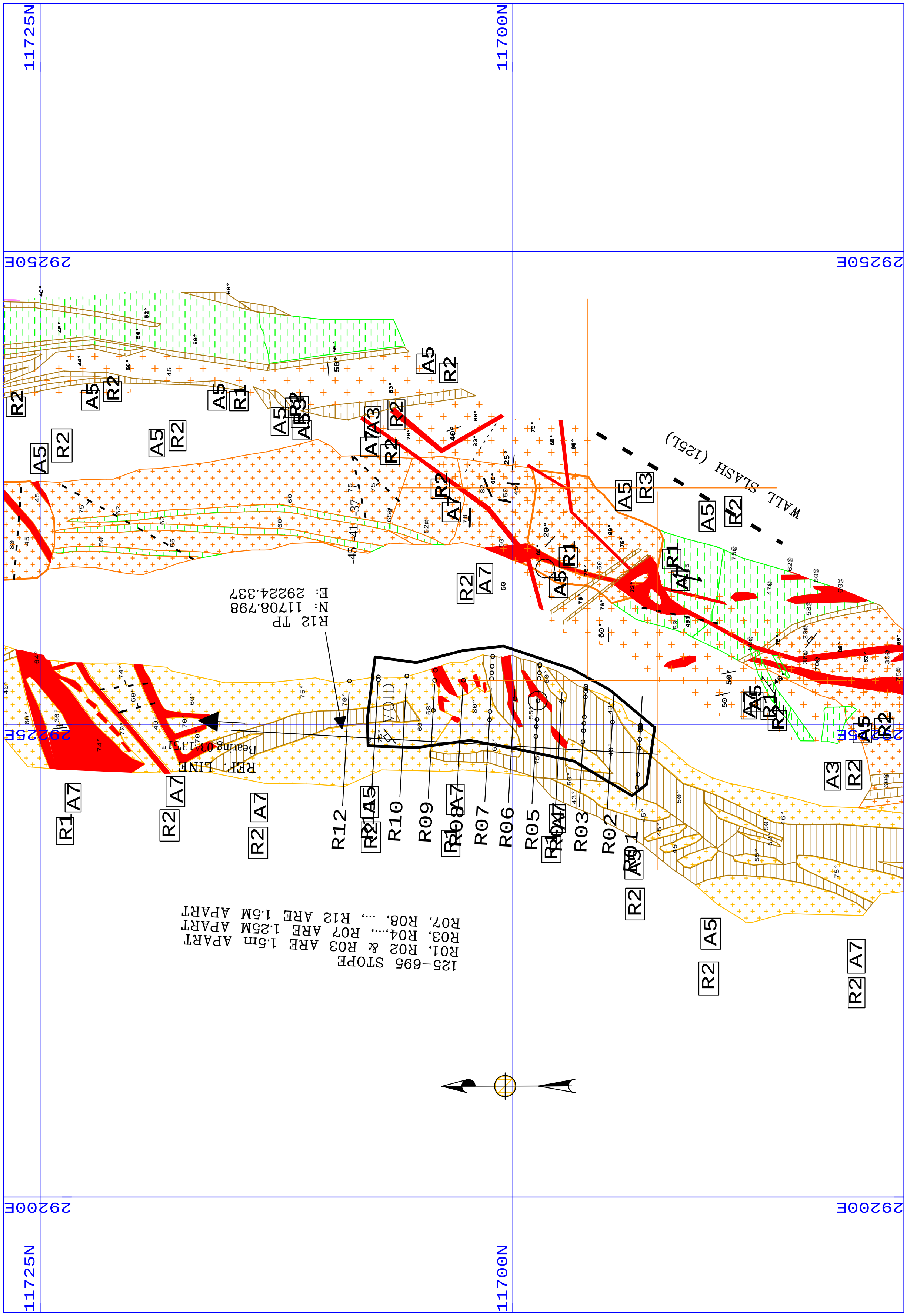
Q' Weighted	RQD	RQD O/C and U/C	Q'
7.4	69.8	62.9	6.7

A	B	C	N'
1	0.2	3.3	4.9

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	125-695	
Modified Stability Number (N')	Sutton N'	1.7
	Parameter Based N'	1.9
	RMR76 to Q (Equation Based Conversion)	4.9
Hydraulic Radius	Actual	6.3
Average HW Overbreak from the Optech:		**3.0** **m**




11725N
11700N
29250E
29225E
29200E
WALL SLASH (125L)
R12 TP
N: 11708.798
E: 29224.337
REF. LINE
VOID
R12
R11
R10
R09
R08
R07
R06
R05
R04
R03
R02
R01
125-695 STOPE
R01, R02 & R03 ARE 1.5m APART
R03, R04,..., R07 ARE 1.25M APART
R07, R08, ..., R12 ARE 1.5M APART

STOPE	150-835	
Strike Length	21.5	
HW Exposed Height	21.2	
Hydraulic Radius	5.3	
Average HW Dip	53.8	°
Percentage of HW that is:		
Gneissic	100	%
Pegmatoidal	0	%
Other	0	%

Sutton's Q' from A/R values

150-835											
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
Gneiss	A7/R1	16	20	320	4	64	0.75	12	10	160	
	A5/R2	5.5	75	412.5	7.5	41.25	1.5	8.25	6	33	
TOTAL:		21.5		732.5		105.25		20.25		193	
				34.1		4.9		0.9		9.0	0.7
UNDERCUT											
Gneiss	A7/R1	16	20	320.0	4.0	64.0	0.8	12.0	10.0	160.0	
	A7/R2	5.5	60	330	6	33	1.5	8.25	6	33	
TOTAL:		21.5		650		97		20.25		193	
				30.2		4.5		0.9		9.0	0.7
									AVERAGE:		0.7

DDH's	RQD in HW:
3402	70
3194	89

Q' Weighted	RQD	Jn	Jr	Ja
1.2	55.8	4.7	0.9	9.0

A	B	C	N'
1.0	0.2	4.5	1.1

150-835

Parameter Based Q' from A/R values

OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
Gneiss	A7/R1	16	75	1200	9	144	0.75	12	4	64	
	A5/R2	5.5	75	412.5	9	49.5	1.5	8.25	3	16.5	
TOTAL:		21.5		1612.5		193.5		20.25		80.5	
				75		9		0.9		3.7	2.1
UNDERCUT											
Gneiss	A7/R1	16	75	1200.0	9.0	144.0	0.8	12.0	4.0	64.0	
	A7/R2	5.5	75	412.5	9	49.5	1.5	8.25	4	22	
TOTAL:		21.5		1612.5		193.5		20.25		86	
				75		9		0.9		4	2.0
									AVERAGE:		2.0

DDH's	RQD in HW:
3402	70
3194	89

Q' Weighted	RQD	Jn	Jr	Ja
2.1	77.3	9.0	0.9	3.9

A	B	C	N'
1.0	0.2	4.5	1.9

150-835			RMR = 9 ln Q' + 44			
OVERCUT		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
Gneiss	A7/R1	16	69	1104	3.6	57.6
	A5/R2	5.5	68	374	8.3	45.65
TOTAL:		21.5		1478		103.25
				68.7		4.8
UNDERCUT						
Gneiss	A7/R1	16	69	1104.0	3.6	57.6
	A7/R2	5.5	73	401.5	5	27.5
TOTAL:		21.5		1505.5		85.1
				70.0		4.0

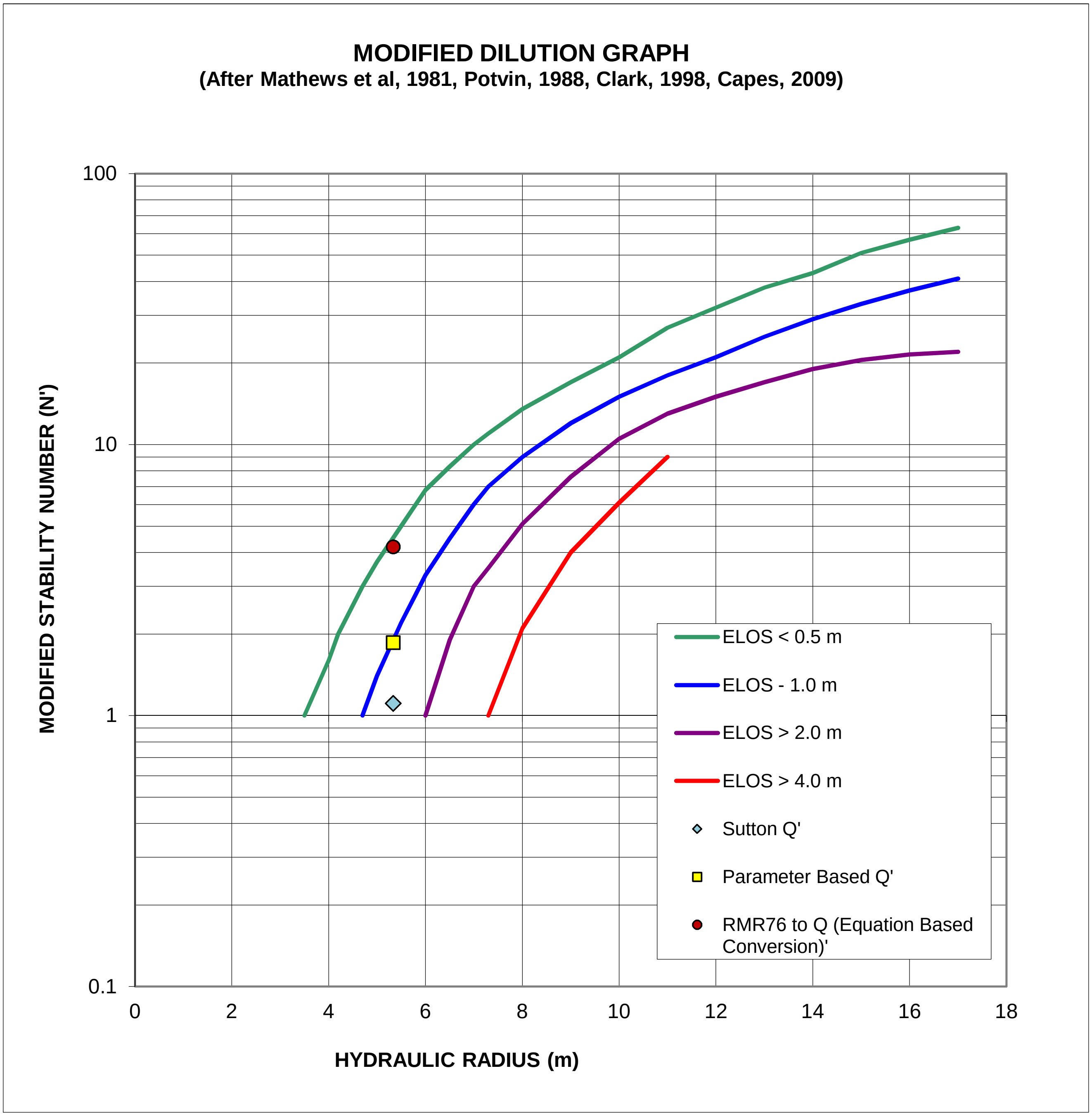
DDH's	RQD in HW:
3402	70
3194	89

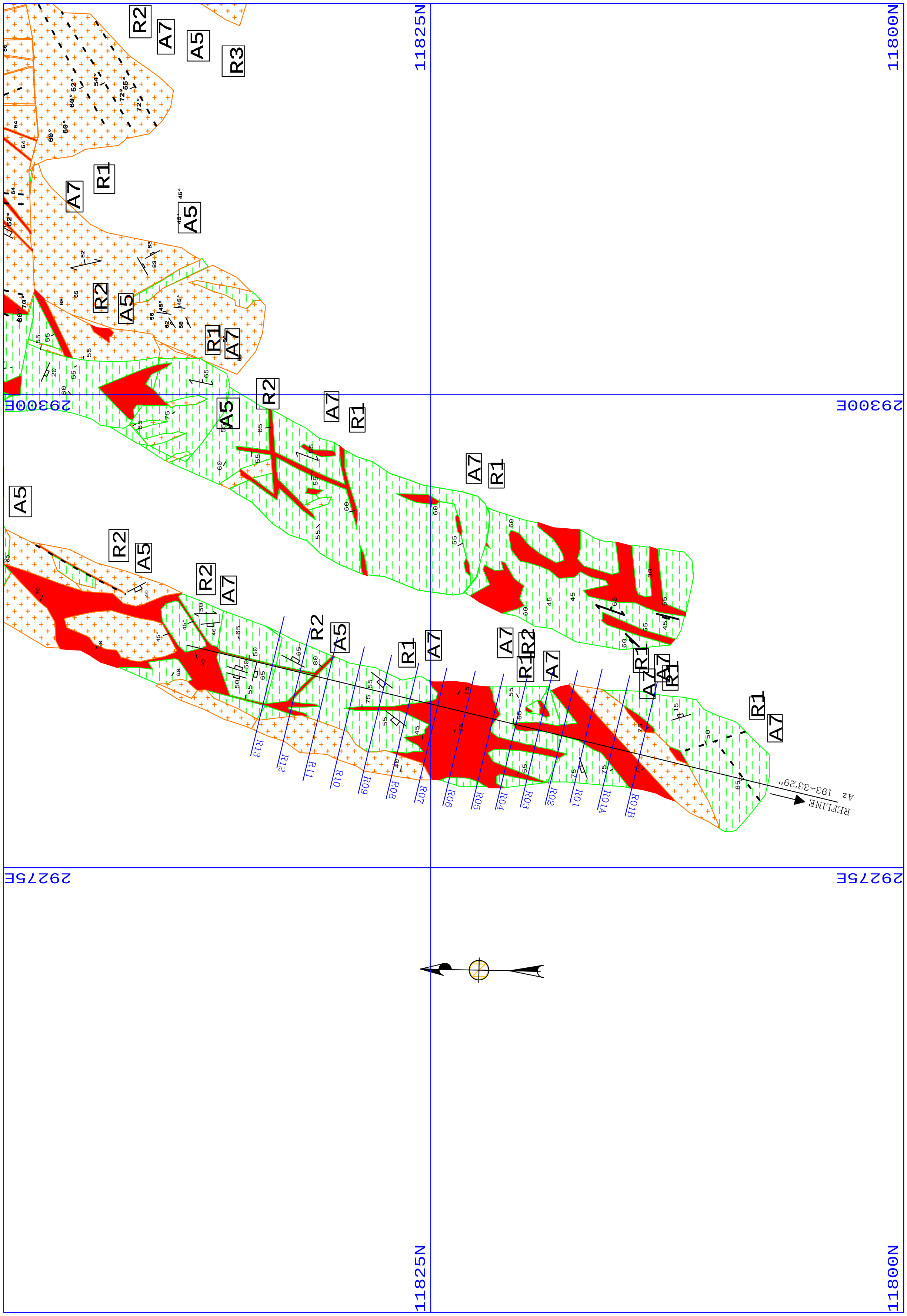
Q' Weighted	RQD	RQD O/C and U/C	Q'
4.7	74.4	69.4	4.4

A	B	C	N'
1.0	0.2	4.5	4.2

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	150-835	
Modified Stability Number (N')	Sutton N'	1.1
	Parameter Based N'	1.9
	RMR76 to Q (Equation Based Conversion)	4.2
Hydraulic Radius	Actual	5.3
Average HW Overbreak from the Optech:		**2.6** m




11825N
11800N
29300E
29275E
A5
A7
R1
R2
R3
R01B
R01A
R01
R02
R03
R04
R05
R06
R07
R08
R09
R10
R11
R12
R13
REFLINE
Az 193°33'29"

STOPE	150-860	
Strike Length	20.5	
HW Exposed Height	18.9	
Hydraulic Radius	4.9	
Average HW Dip	62.9	°
Percentage of HW that is:		
Gneissic	7	%
Pegmatoidal	93	%
Other	0	%

Sutton's Q' from A/R values

150-860												
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
Pegmatoid	A5/R2	4.5	75	337.5	7.5	33.75	1.5	6.75	6	27	
	A7/R2	6.7	60	402	6	40.2	1.5	10.05	6	40.2	
	A7/R1	9.3	20	186	4	37.2	0.75	6.975	10	93	
TOTAL:		20.5		925.5		111.2		23.8		160.2	
				45.1		5.4		1.2		7.8	1.2
UNDERCUT											
Pegmatoid	A5/R2	14.5	75	1087.5	7.5	108.75	1.5	21.75	6	87	
	A7/R1	3	20	60	4	12	0.75	2.25	10	30	
Gneiss	A5/R2	3	75	225.0	7.5	22.5	1.5	4.5	6.0	18.0	
TOTAL:		20.5		1372.5		143.3		28.5		135.0	
				67.0		7.0		1.4		6.6	2.0
									AVERAGE:		1.6

DDH's	RQD in HW:
3163	57

Q' Weighted	RQD	Jn	Jr	Ja
1.6	56.4	6.2	1.3	7.2

A	B	C	N'
1	0.2	5.3	1.7

150-860

Parameter Based Q' from A/R values

OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
Pegmatoid	A5/R2	4.5	60	270	7.5	33.75	1.5	6.75	4	18	
	A7/R2	6.7	50	335	6	40.2	1.5	10.05	6	40.2	
	A7/R1	9.3	25	232.5	4	37.2	0.75	6.975	10	93	
TOTAL:		20.5		837.5		111.15		23.8		151.2	
				40.9		5.4		1.2		7.4	1.2
UNDERCUT											
Pegmatoid	A5/R2	14.5	60	870	7.5	108.75	1.5	21.75	4	58	
	A7/R1	3	25	75	4	12	0.75	2.25	10	30	
Gneiss	A5/R2	3	75	225	9	27	1.5	4.5	3	9	
TOTAL:		20.5		1170.0		147.8		28.5		97.0	
				57.1		7.2		1.4		4.7	2.3
									AVERAGE:		1.8

DDH's	RQD in HW:
3163	57

Q' Weighted	RQD	Jn	Jr	Ja
1.7	51.6	6.3	1.3	6.1

A	B	C	N'
1	0.2	5.3	1.8

150-860

RMR = 9 ln Q' + 44

OVERCUT		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
Pegmatoid	A5/R2	4.5	63	283.5	2.5	11.25
	A7/R2	6.7	73	489.1	4.7	31.49
	A7/R1	9.3	38	353.4	0.3	2.79
TOTAL:		20.5		1126.0		45.5
				54.9		2.2

UNDERCUT						
Pegmatoid	A5/R2	14.5	63	913.5	2.5	36.25
	A7/R1	3	38	114	0.3	0.9
Gneiss	A5/R2	3	68	204.0	8.3	24.9
TOTAL:		20.5		1231.5		62.1
				60.1		3.0

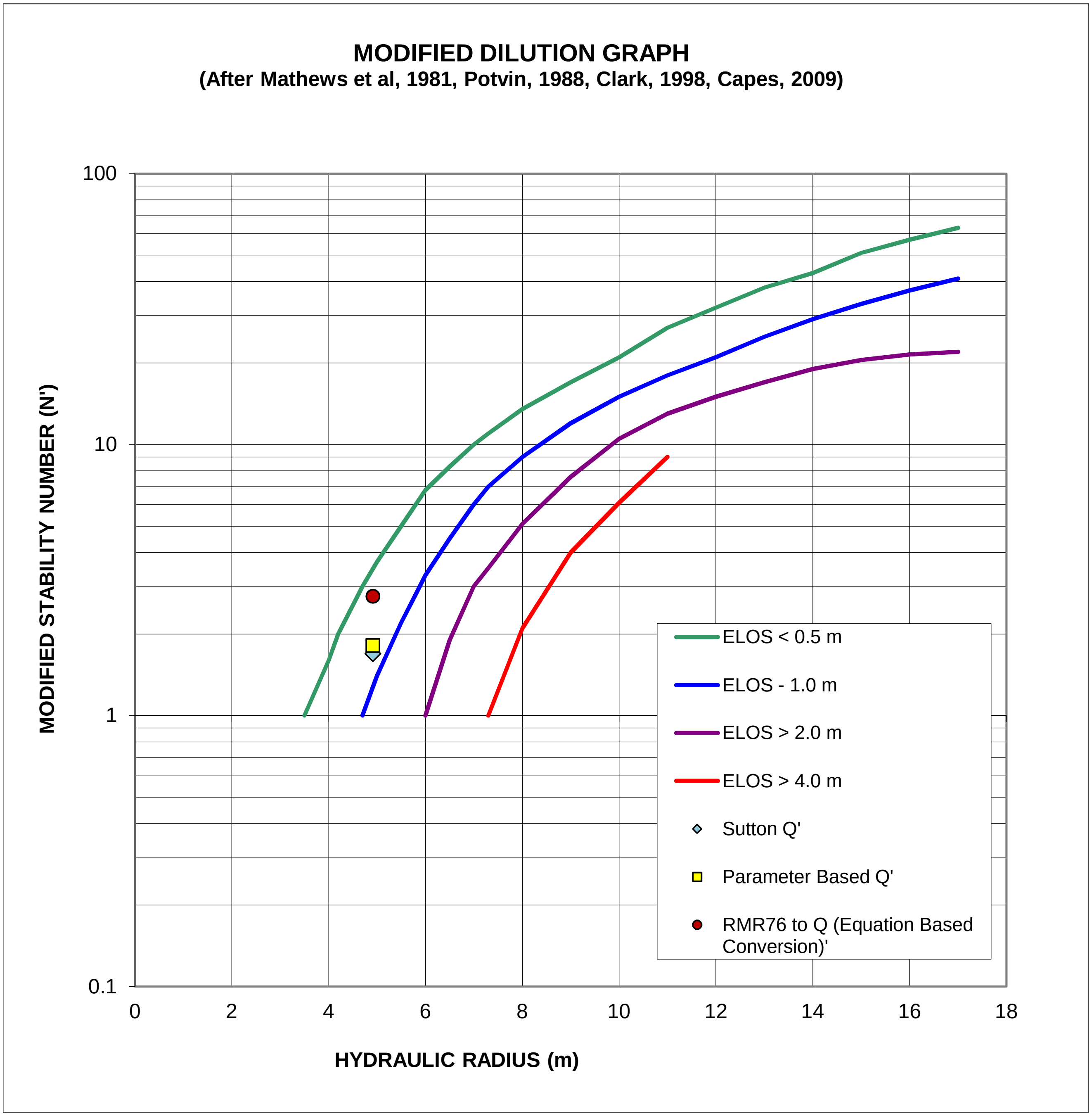
DDH's	RQD in HW:
3163	57

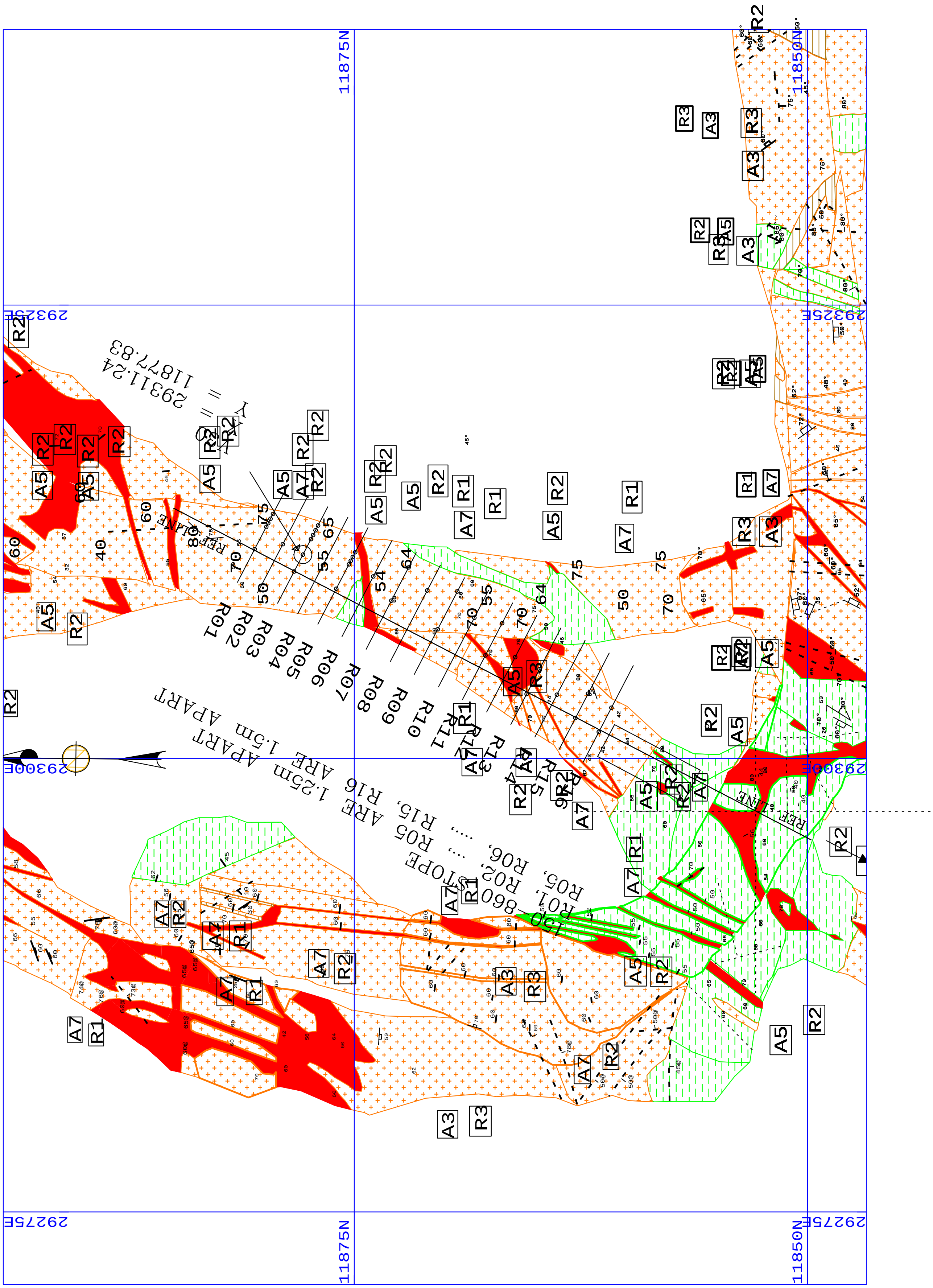
Q' Weighted	RQD	RQD O/C and U/C	Q'
2.6	57.3	57.5	2.6

A	B	C	N'
1	0.2	5.3	2.8

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	150-860	
Modified Stability Number (N')	Sutton N'	1.7
	Parameter Based N'	1.8
	RMR76 to Q (Equation Based Conversion)	2.8
Hydraulic Radius	Actual	4.9
Average HW Overbreak from the Optech:		**2.8** m




STOPE 860-150
R01, R02, ..., R05 ARE 1.25m APART
R05, R06, ..., R15, R16 ARE 1.5m APART
REF LINE
X = 29311.24
Y = 11877.83
11875N
11850N
29325E
29300E
29275E

STOPE	150-870	
Strike Length	23	
HW Exposed Height	20.5	
Hydraulic Radius	5.4	
Average HW Dip	56.1	°
Percentage of HW that is:		
Gneissic	0	%
Pegmatoidal	100	%
Other	0	%

Sutton's Q' from A/R values

150-870											
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
PEGMATOID	A7/R1	6	20	120	4	24	0.75	4.5	10	60	
	A5/R2	17	75	1275	7.5	127.5	1.5	25.5	6	102	
TOTAL:		23		1395		151.5		30		162	
				60.7		6.6		1.3		7.0	1.7
UNDERCUT											
PEGMATOID	A7/R1	5	20	100.0	4.0	20.0	0.8	3.8	10.0	50.0	
	A5/R1	8	50	400	4	32	0.75	6	8	64	
	A5/R2	10	75	750	7.5	75	1.5	15	6	60	
TOTAL:		23		1250.0		127.0		24.8		174.0	
				54.3		5.5		1.1		7.6	1.4
									AVERAGE:		1.6

DDH's	RQD in HW:
3138	45
3137	63
3129	74
3128.0	70.0

Q' Weighted	RQD	Jn	Jr	Ja
1.6	61.2	6.1	1.2	7.3

A	B	C	N'
1.0	0.2	4.7	1.5

150-870

Parameter Based Q' from A/R values

		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
OVERCUT											
PEGMATOID	A7/R1	6	25	150	4	24	0.75	4.5	10	60	
	A5/R2	17	60	1020	7.5	127.5	1.5	25.5	4	68	
TOTAL:		23		1170		151.5		30		128	
				50.86956522		6.586956522		1.3		5.6	1.8
UNDERCUT											
PEGMATOID	A7/R1	5	25	125.0	4.0	20.0	0.8	3.8	10.0	50.0	
	A5/R1	8	40	320	4	32	0.75	6	8	64	
	A5/R2	10	60	600	7.5	75	1.5	15	4	40	
TOTAL:		23		1045		127		24.8		154	
				45.4		5.5		1.1		6.7	1.3
									AVERAGE:		1.6

DDH's	RQD in HW:
3138	45
3137	63
3129	74
3128.0	70.0

Q' Weighted	RQD	Jn	Jr	Ja
1.9	58.1	6.1	1.2	6.1

A	B	C	N'
1.0	0.2	4.7	1.7

RMR = 9 ln Q' + 44

150-870						
OVERCUT		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
PEGMATOID	A7/R1	6	38	228	0.3	1.8
	A5/R2	17	63	1071	2.5	42.5
TOTAL:		23		1299		44.3
				56.5		1.9
UNDERCUT						
PEGMATOID	A7/R1	5	38	190.0	0.3	1.5
	A5/R1	8	25	200	0.3	2.4
	A5/R2	10	63	630	2.5	25
TOTAL:		23		1020.0		28.9
				44.3		1.3

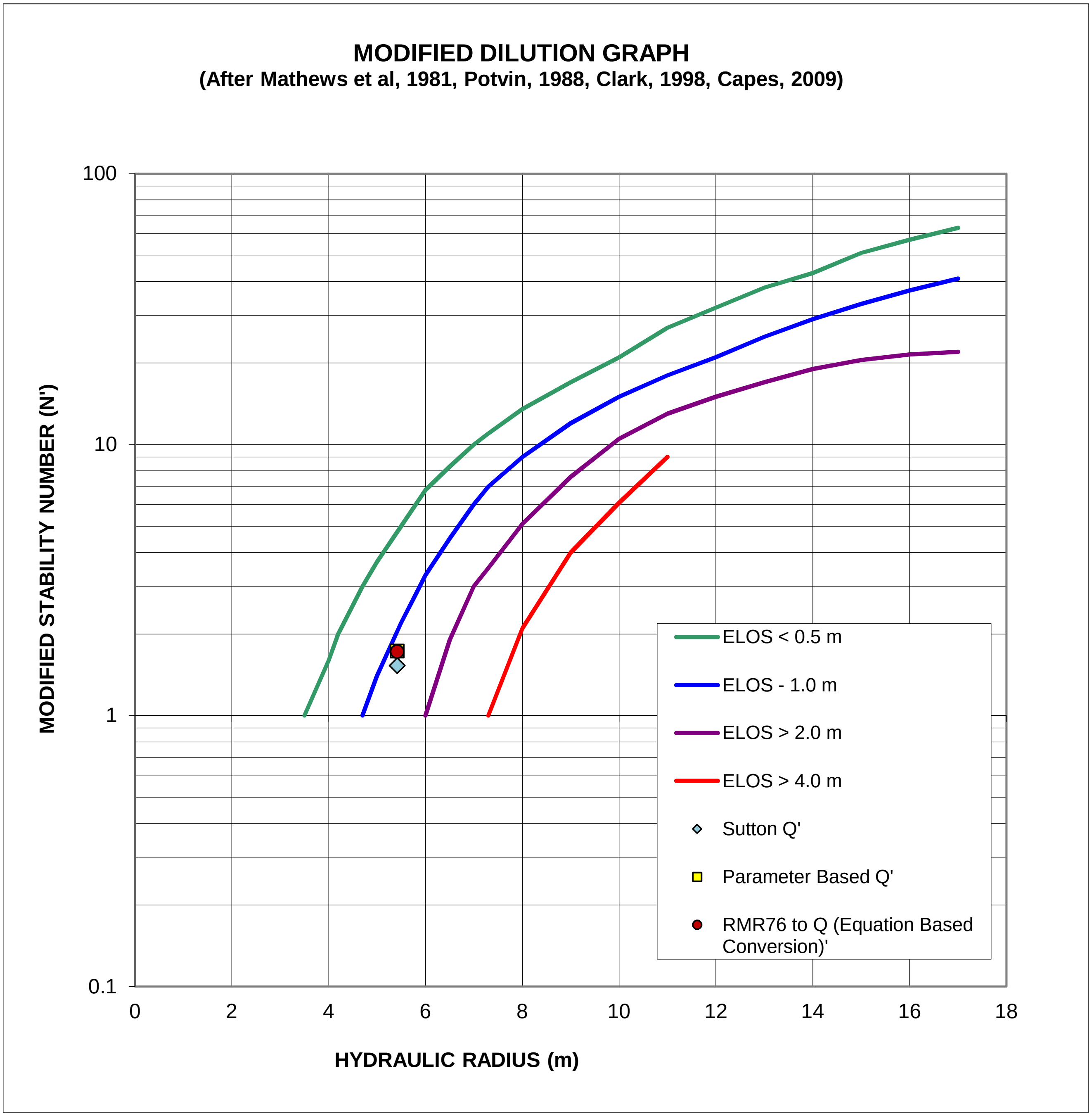
DDH's	RQD in HW:
3138	45
3137	63
3129	74
3128.0	70.0

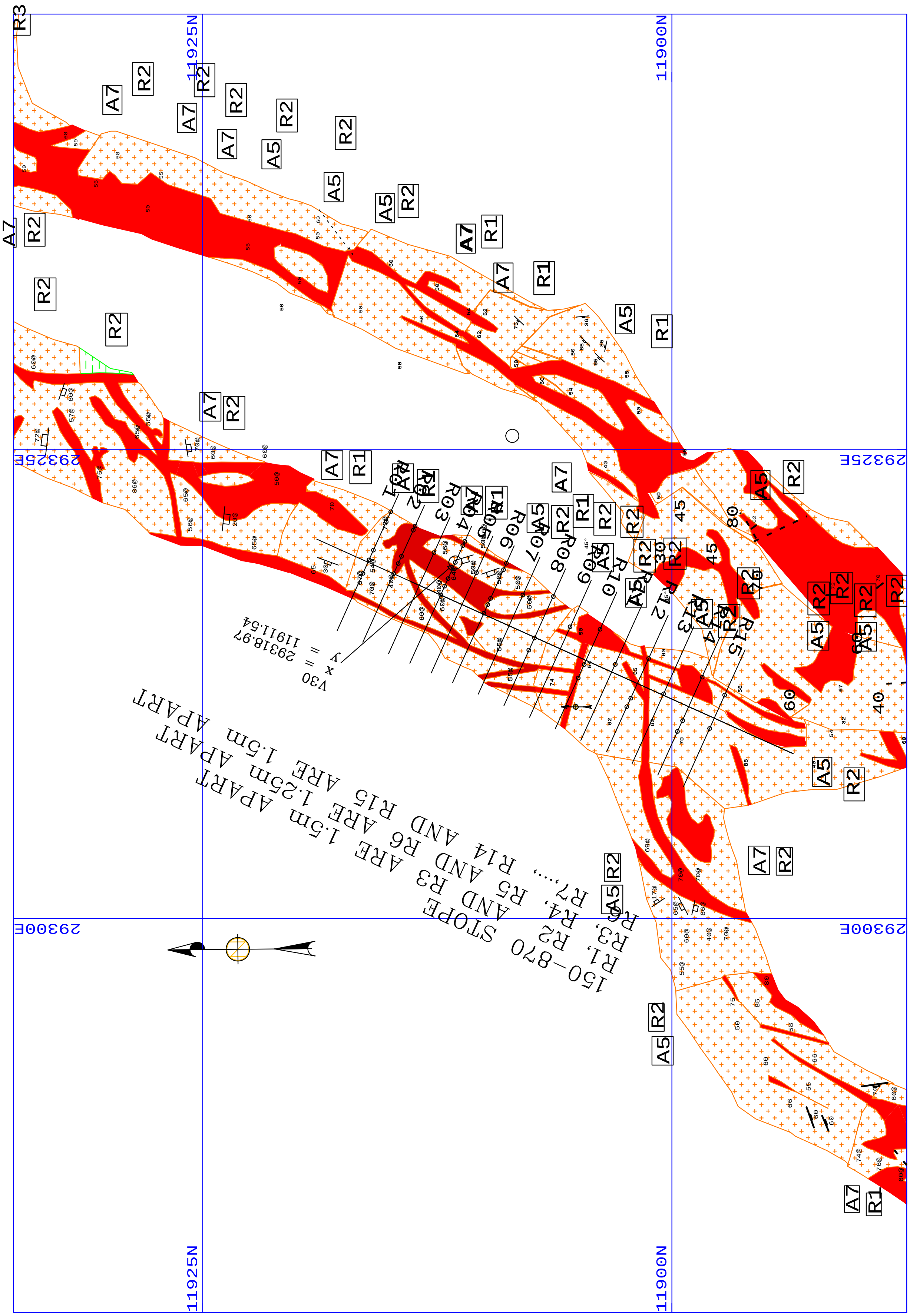
Q' Weighted	RQD	RQD O/C and U/C	Q'
1.9	58.8	50.4	1.6

A	B	C	N'
1.0	0.2	4.7	1.7

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	150-870		
Modified Stability Number (N')	Sutton N'	1.5	
	Parameter Based N'	1.7	
	RMR76 to Q (Equation Based Conversion)	1.7	
Hydraulic Radius	Actual	5.4	
Average HW Overbreak from the Optech:		**3.0**	**m**




150-870 STOPE
R1, R2 AND R3 ARE 1.5m APART
R4, R5 AND R6 ARE 1.25m APART
R7,... R14 AND R15 ARE 1.5m APART
V30
x = 29318.97
y = 11911.54
11925N
11900N
29325E
29300E

STOPE	150-880	
Strike Length	20.5	
HW Exposed Height	19.5	
Hydraulic Radius	5.0	
Average HW Dip	59.7	°
Percentage of HW that is:		
Gneissic	15	%
Pegmatoidal	85	%
Other	0	%

150-880			Sutton's Q' from A/R values								
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
Pegmatoid	A3/R3	5	100	500	9	45	2.3	11.5	1	5	
	A7/R2	10.5	60	630	6	63	1.5	15.75	6	63	
	A7/R1	5	20	100	4	20	0.75	3.75	10	50	
TOTAL:		20.5		1230.0		128.0		31.0		118.0	
				60.0		6.2		1.5		5.8	2.5
UNDERCUT											
Pegmatoid	A7/R1	14.5	20	290	4	58	0.75	10.875	10	145	
Gneiss	A7/R1	6	20	120	4	24	0.75	4.5	10	60	
TOTAL:		20.5		410.0		82.0		15.4		205.0	
				20.0		4.0		0.8		10.0	0.4
									AVERAGE:		1.4

DDH's	RQD in HW:
3153	87

Q' Weighted	RQD	Jn	Jr	Ja
1.6	55.7	5.1	1.1	7.9

A	B	C	N'
1.0	0.2	5.0	1.6

Parameter Based Q' from A/R values

150-880											
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
Pegmatoid	A3/R3	5	75	375	9	45	2.3	11.5	1	5	
	A7/R2	10.5	50	525	6	63	1.5	15.75	6	63	
	A7/R1	5	25	125	4	20	0.75	3.75	10	50	
TOTAL:		20.5		1025		128		31.0		118.0	
				50.0		6.2		1.5		5.8	2.1
UNDERCUT											
Pegmatoid	A7/R1	14.5	25	362.5	4	58	0.75	10.875	10	145	
Gneiss	A7/R1	6	75	450	9	54	0.75	4.5	4	24	
TOTAL:		20.5		812.5		112		15.4		169	
				39.6		5.5		0.8		8.2	0.7
									AVERAGE:		1.4

DDH's	RQD in HW:
3153	87

Q' Weighted	RQD	Jn	Jr	Ja
1.6	58.9	5.9	1.1	7.0

A	B	C	N'
1.0	0.2	5.0	1.6

150-880

RMR = 9 ln Q' + 44

OVERCUT		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
Pegmatoid	A3/R3	5	62	310	6.2	31
	A7/R2	10.5	73	766.5	4.7	49.35
	A7/R1	5	38	190	0.3	1.5
TOTAL:		20.5		1266.5		81.9
				61.8		4.0

UNDERCUT						
Pegmatoid	A7/R1	14.5	38	551	0.3	4.35
Gneiss	A7/R1	6	69	414	3.6	21.6
TOTAL:		20.5		965.0		26.0
				47.1		1.3

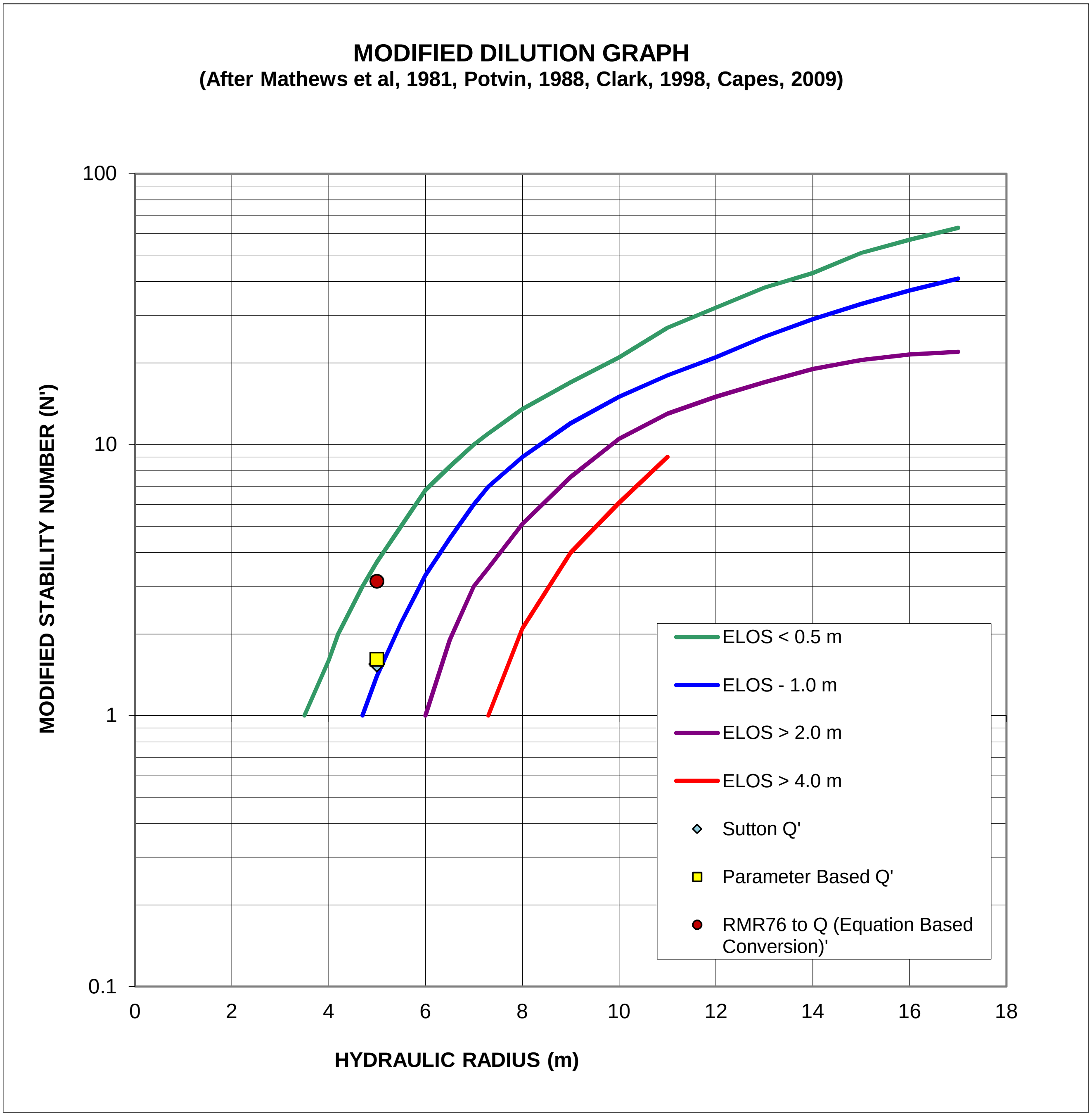
DDH's	RQD in HW:
3153	87

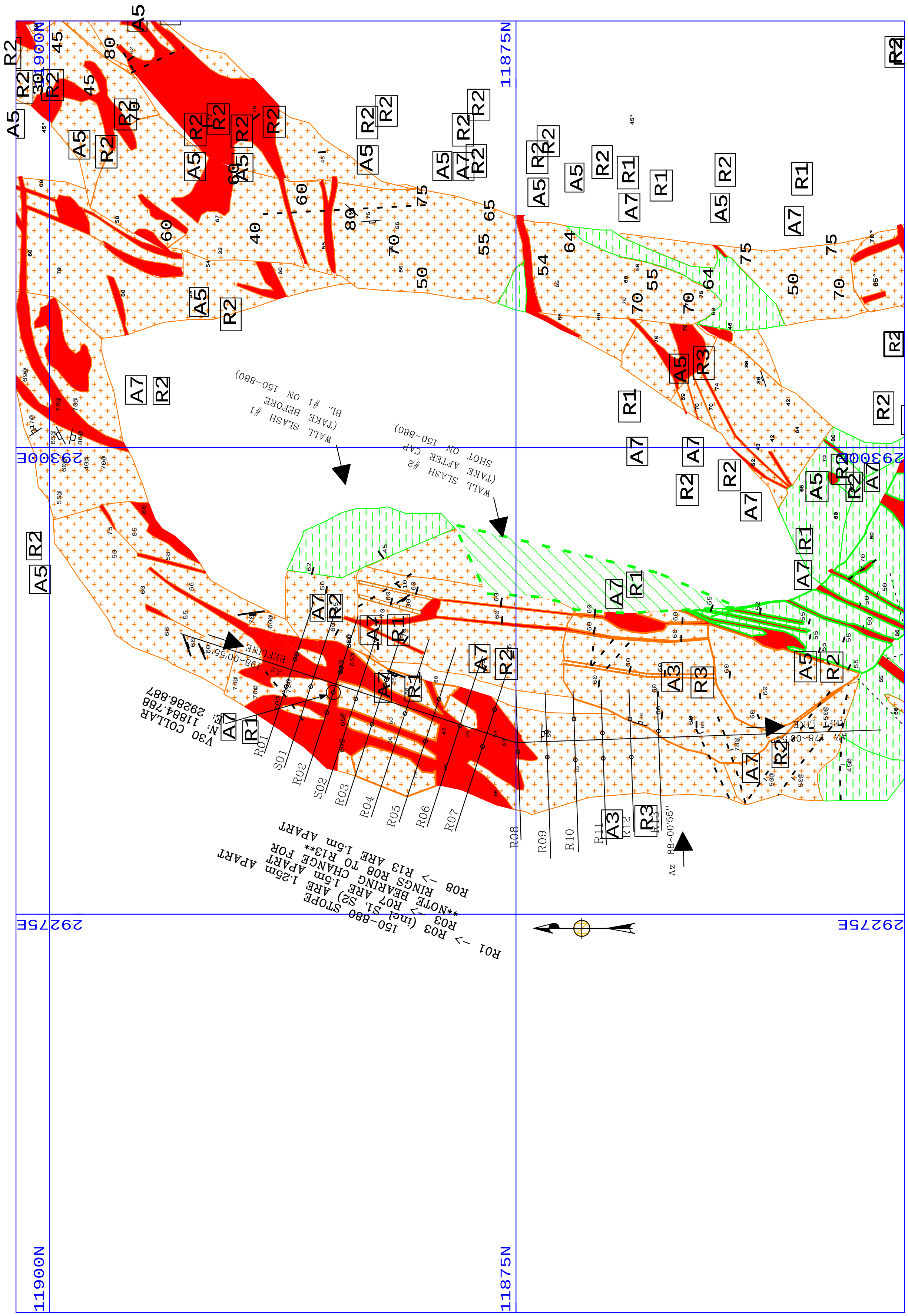
Q' Weighted	RQD	RQD O/C and U/C	Q'
3.2	65.3	54.4	2.6

A	B	C	N'
1.0	0.2	5.0	3.1

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	150-880	
Modified Stability Number (N')	Sutton N'	1.6
	Parameter Based N'	1.6
	RMR76 to Q (Equation Based Conversion)	3.1
Hydraulic Radius	Actual	5.0
Average HW Overbreak from the Optech:		**1.9** m





11900N
11875N
29300E
29275E

STOPE	150-900	
Strike Length	27	
HW Exposed Height	21.7	
Hydraulic Radius	6.0	
Average HW Dip	52.3	°
Percentage of HW that is:		
Gneissic	6	%
Pegmatoidal	94	%
Other	0	%

150-900		Sutton's Q' from A/R values									
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
PEGMATOID	A7/R1	7.6	20	152	4	30.4	0.75	5.7	10	76	
	A7/R2	10.3	60	618	6	61.8	1.5	15.45	6	61.8	
	A5/R2	6.1	75	457.5	7.5	45.75	1.5	9.15	6	36.6	
BQFG	A7/R2	3	60	180.0	6	18.0	1.5	4.5	6.0	18.0	
TOTAL:		27		1407.5		156.0		34.8		192.4	
				52.1		5.8		1.3		7.1	1.6
UNDERCUT											
PEGMATOID	A7/R1	3.5	20	70	4	14	0.75	2.625	10	35	
	A7/R2	13	60	780.0	6	78.0	1.5	19.5	6.0	78.0	
	A5/R2	10.5	75	787.5	7.5	78.8	1.5	15.8	6.0	63.0	
TOTAL:		27		1637.5		170.8		37.9		176.0	
				60.6		6.3		1.4		6.5	2.1
									AVERAGE:		1.8

DDH's	RQD in HW:
3126	87
3298	100
3436	46

Q' Weighted	RQD	Jn	Jr	Ja
2.3	69.2	6.1	1.3	6.8

A	B	C	N'
1	0.2	4.3	2.0

150-900

Parameter Based Q' from A/R values

OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
PEGMATOID	A7/R1	7.6	25	190	4	30.4	0.75	5.7	10	76	
	A7/R2	10.3	50	515	6	61.8	1.5	15.45	6	61.8	
	A5/R2	6.1	60	366	7.5	45.75	1.5	9.15	4	24.4	
BQFG	A7/R2	3	75	225	9	27	1.5	4.5	4	12.0	
TOTAL:		27		1296.0		165.0		34.8		174.2	
				48		6.1		1.3		6.5	1.6
UNDERCUT											
PEGMATOID	A7/R1	3.5	25	87.5	4	14	0.75	2.625	10	35	
	A7/R2	13	50	650	6	78	1.5	19.5	6	78	
	A5/R2	10.5	60	630.0	7.5	78.8	1.5	15.8	4.0	42.0	
TOTAL:		27		1367.5		170.8		37.9		155.0	
				50.6		6.3		1.4		5.7	2.0
									AVERAGE:		1.8

DDH's	RQD in HW:
3126	87
3298	100
3436	46

Q' Weighted	RQD	Jn	Jr	Ja
2.4	66.3	6.2	1.3	6.1

A	B	C	N'
1	0.2	4.3	2.0

150-900

RMR = 9 ln Q' + 44

OVERCUT		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
PEGMATOID	A7/R1	7.6	38	288.8	0.3	2.28
	A7/R2	10.3	73	751.9	4.7	48.41
	A5/R2	6.1	63	384.3	2.5	15.25
BQFG	A7/R2	3	73	219.0	5	15.0
TOTAL:		27		1644.0		80.9
				60.9		3.0

UNDERCUT		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
PEGMATOID	A7/R1	3.5	38	133	0.3	1.05
	A7/R2	13	73	949.0	4.7	61.1
	A5/R2	10.5	63	661.5	2.5	26.3
TOTAL:		27		1743.5		88.4
				64.6		3.3

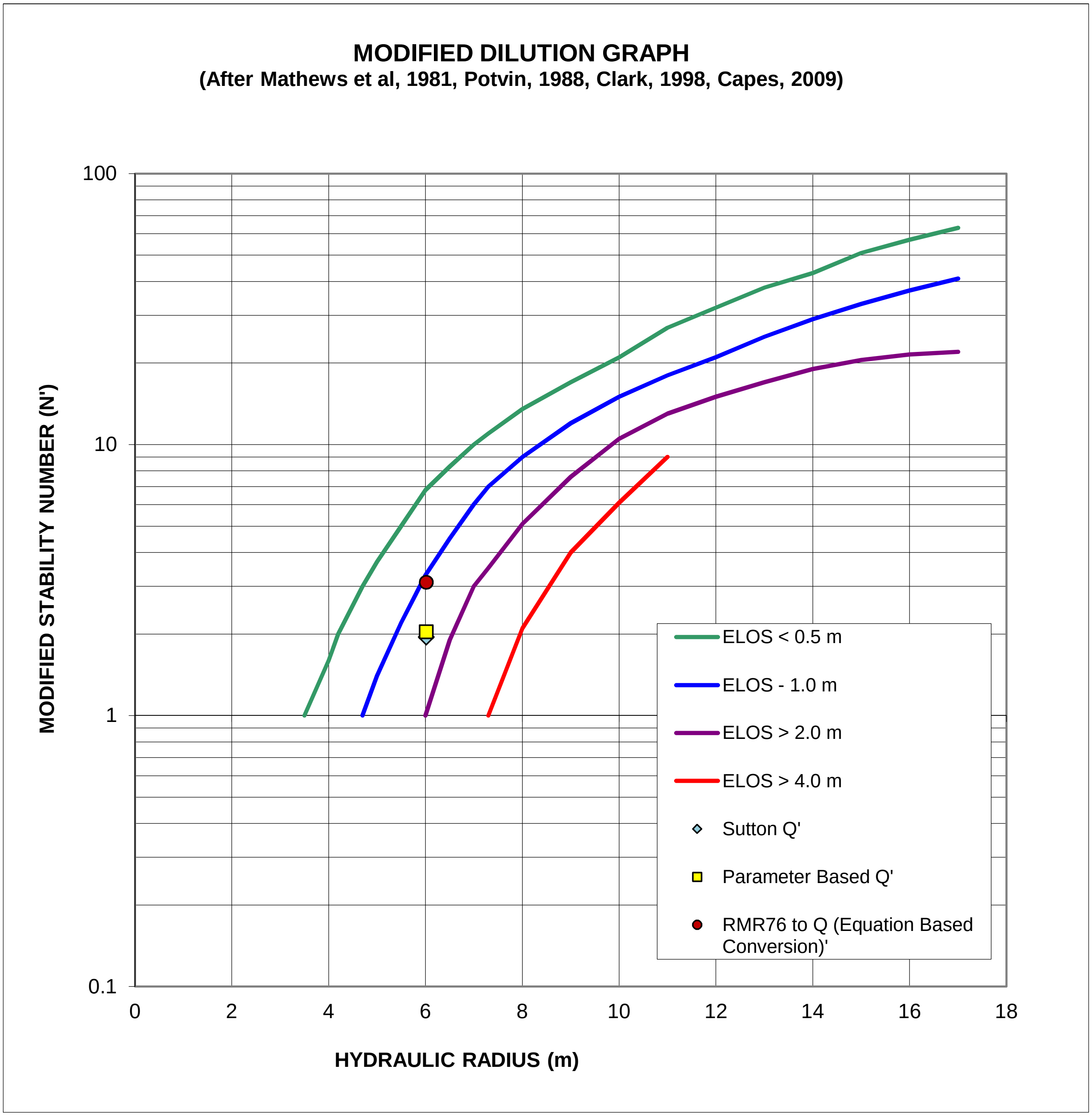
DDH's	RQD in HW:
3126	87
3298	100
3436	46

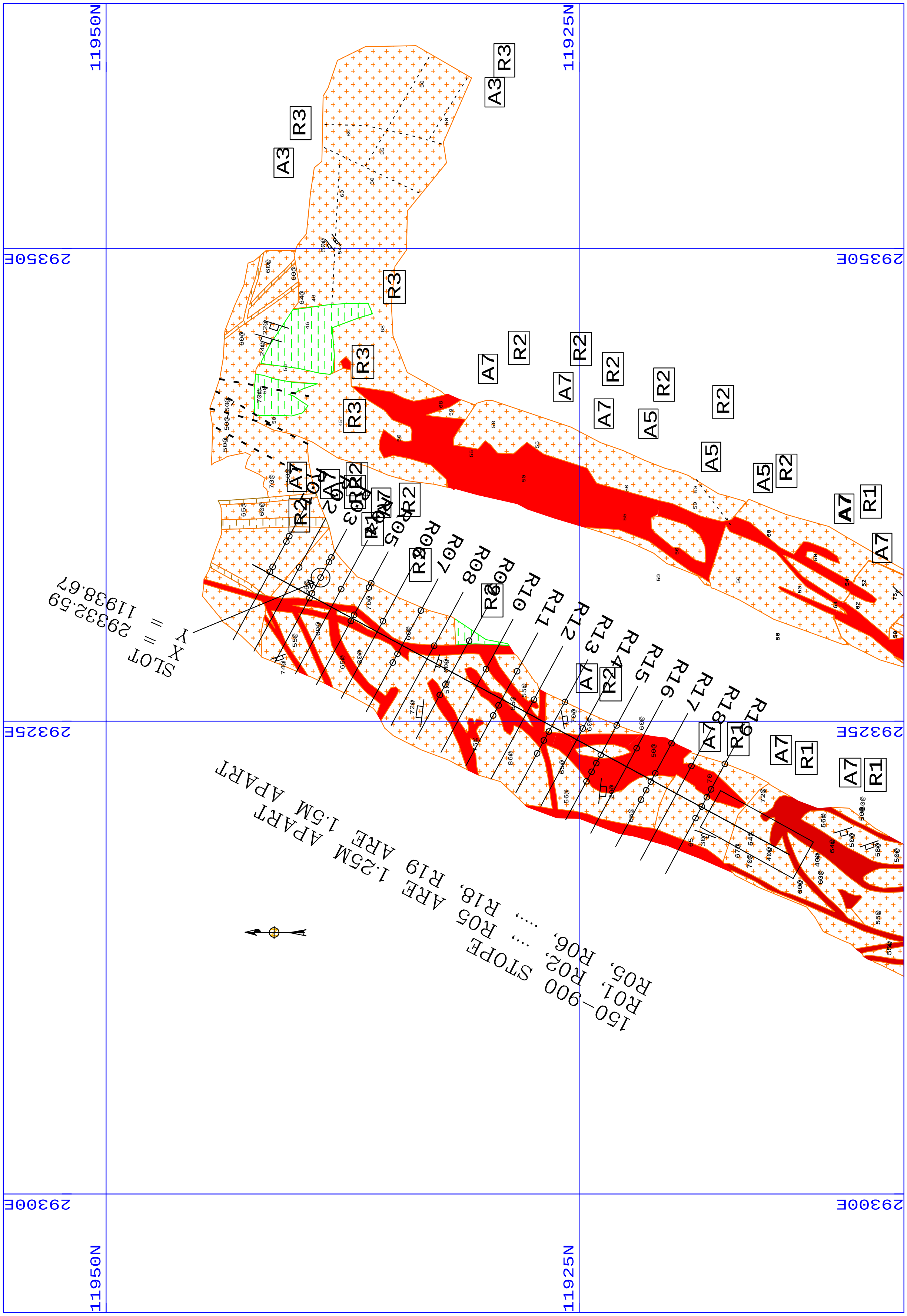
Q' Weighted	RQD	RQD O/C and U/C	Q'
3.6	71.7	62.7	3.1

A	B	C	N'
1	0.2	4.3	3.1

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	150-900	
Modified Stability Number (N')	Sutton N'	2.0
	Parameter Based N'	2.0
	RMR76 to Q (Equation Based Conversion)	3.1
Hydraulic Radius	Actual	6.0
Average HW Overbreak from the Optech:		**1.6** m




11950N
11925N
29350E
29325E
29300E
150-900 STOPE
R01, R02, ... R05 ARE 1.25M APART
R05, R06, ... R18, R19 ARE 1.5M APART
SLOT
X = 29332.59
Y = 11938.67

STOPE	170-860	
Strike Length	32	
HW Exposed Height	29	
Hydraulic Radius	7.6	
Average HW Dip	49.7	°
Percentage of HW that is:		
Gneissic	4	%
Pegmatoidal	96	%
Other	0	%

170-860			Sutton's Q' from A/R values								
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
BQFG	A5/R2	2.5	75	187.5	7.5	18.75	1.5	3.75	6	15	
Pegmatoid	A5/R2	29.5	75	2212.5	7.5	221.25	1.5	44.25	6	177	
TOTAL:		32		2400		240		48		192	
				75.0		7.5		1.5		6.0	2.5
UNDERCUT											
Pegmatoid	A7/R2	3.5	60	210.0	6.0	21.0	1.5	5.3	6.0	21.0	
	A7/R1	26.5	20	530	4	106	0.75	19.875	10	265	
	A5/R1	2	50	100	4	8	0.75	1.5	8	16	
TOTAL:		32		840.0		135.0		26.6		302.0	
				26.3		4.2		0.8		9.4	0.5

DDH's	RQD in HW:
3164	60
3154	77
3145	77

Q' Weighted	RQD	Jn	Jr	Ja
2	63	5.9	1.2	7.7

A	B	C	N'
1.0	0.2	4.1	1.3

170-860

Parameter Based Q' from A/R values

OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
BQFG	A5/R2	2.5	75	187.5	9	22.5	1.5	3.75	3	7.5	
Pegmatoid	A5/R2	29.5	60	1770	7.5	221.25	1.5	44.25	4	118	
TOTAL:		32		1957.5		243.75		48		125.5	
				61.2		7.6		1.5		3.9	3.1
UNDERCUT											
Pegmatoid	A7/R2	3.5	50	175.0	6.0	21.0	1.5	5.3	6.0	21.0	
	A7/R1	26.5	25	662.5	4	106	0.75	19.875	10	265	
	A5/R1	2	40	80	4	8	0.75	1.5	8	16	
TOTAL:		32		917.5		135		26.6		302	
				28.7		4.2		0.8		9.4	0.6

DDH's	RQD in HW:
3164	60
3154	77
3145	77

Q' Weighted	RQD	Jn	Jr	Ja
2	61	5.9	1.2	6.7

A	B	C	N'
1.0	0.2	4.1	1.5

170-860

Parameter Based Q' from A/R values

OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
BQFG	A5/R2	2.5	75	187.5	9	22.5	1.5	3.75	3	7.5	
Pegmatoid	A5/R2	29.5	60	1770	7.5	221.25	1.5	44.25	4	118	
TOTAL:		32		1957.5		243.75		48		125.5	
				61.2		7.6		1.5		3.9	3.1
UNDERCUT											
Pegmatoid	A7/R2	3.5	50	175.0	6.0	21.0	1.5	5.3	6.0	21.0	
	A7/R1	26.5	25	662.5	4	106	0.75	19.875	10	265	
	A5/R1	2	40	80	4	8	0.75	1.5	8	16	
TOTAL:		32		917.5		135		26.6		302	
				28.7		4.2		0.8		9.4	0.6

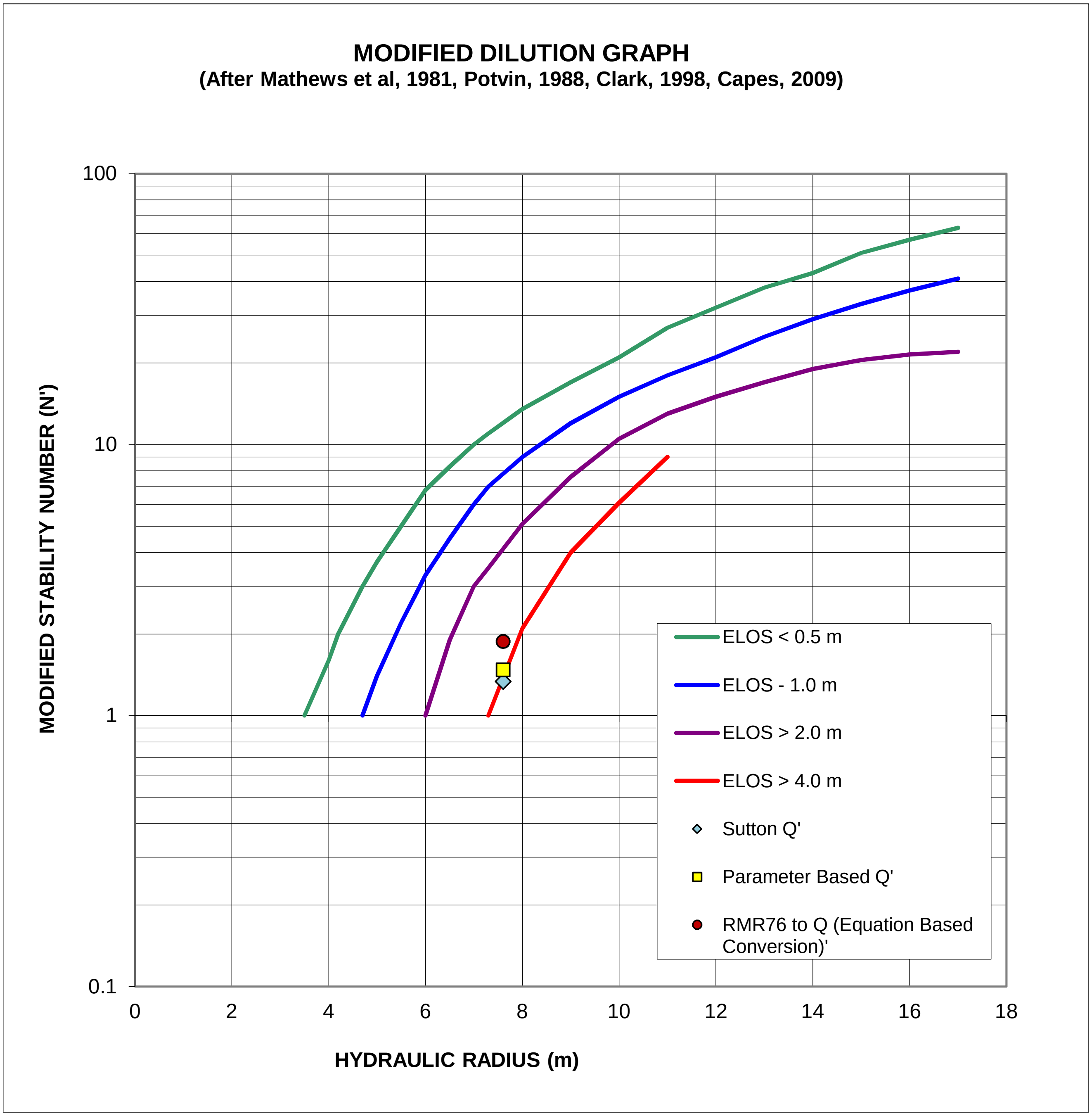
DDH's	RQD in HW:
3164	60
3154	77
3145	77

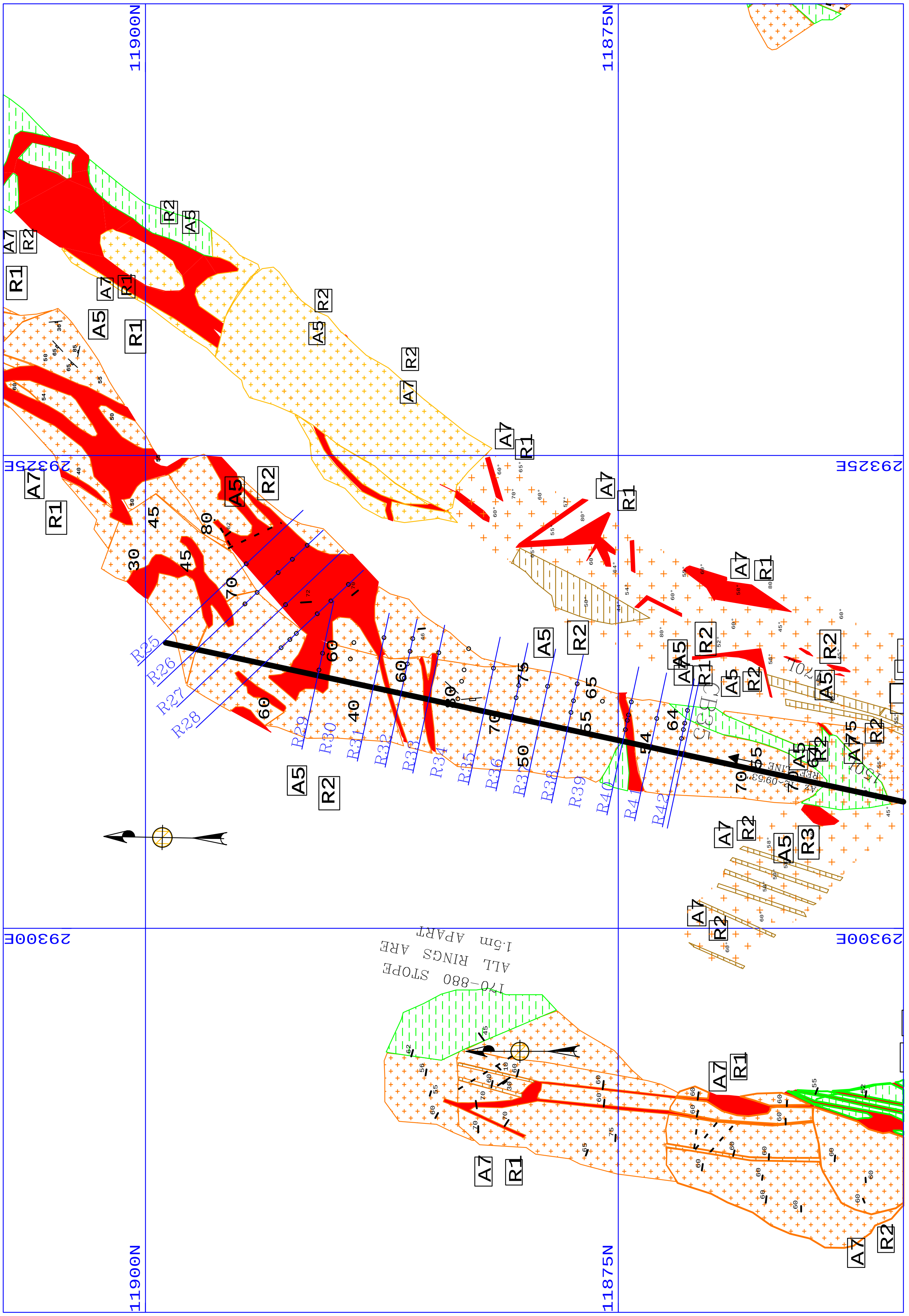
Q' Weighted	RQD	Jn	Jr	Ja
2	61	5.9	1.2	6.7

A	B	C	N'
1.0	0.2	4.1	1.5

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	170-860		
Modified Stability Number (N')	Sutton N'	1.3	
	Parameter Based N'	1.5	
	RMR76 to Q (Equation Based Conversion)	1.9	
Hydraulic Radius	Actual	7.6	
Average HW Overbreak from the Optech:		**5.4**	**m**





11900N
11875N
29325E
29300E
170-880 STOPE
ALL RINGS ARE
1.5m APART
R25
R26
R27
R28
R29
R30
R31
R32
R33
R34
R35
R36
R37
R38
R39
R40
R41
R42
A5
A7
R1
R2
R3

STOPE	170-870	
Strike Length	16	
HW Exposed Height	29.6	
Hydraulic Radius	5.2	
Average HW Dip	49.2	°
Percentage of HW that is:		
Gneissic	9	%
Pegmatoidal	91	%
Other	0	%

Sutton's Q' from A/R values

170-870											
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
PEGMATOID	A5/R2	6	75	450	7.5	45	1.5	9	6	36	
	A5/R1	6	50	300	4	24	0.75	4.5	8	48	
	A7/R1	4	20	80	4	16	0.75	3	10	40	
TOTAL:		16		830.0		85.0		16.5		124.0	
				51.9		5.3		1.0		7.8	1.3
UNDERCUT											
BQFG	A5/R2	3	75	225	7.5	22.5	1.5	4.5	6	18	
PEGMATOID	A5/R2	9.5	75	712.5	7.5	71.25	1.5	14.25	6	57	
	A7/R2	3.5	60	210.0	6	21.0	1.5	5.3	6.0	21.0	
TOTAL:		16		1147.5		114.8		24.0		96.0	
				71.7		7.2		1.5		6.0	2.5
									AVERAGE:		1.9

DDH's	RQD in HW:
3140	72
3139	97
3446	63
3167	78
3145.0	77.0
3146.0	87.0

Q' Weighted	RQD	Jn	Jr	Ja
2.2	74.7	6.2	1.3	6.9

A	B	C	N'
1	0.2	4.1	1.8

170-870

Parameter Based Q' from A/R values

		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
OVERCUT											
PEGMATOID	A5/R2	6	60	360	7.5	45	1.5	9	4	24	
	A5/R1	6	40	240	4	24	0.75	4.5	8	48	
	A7/R1	4	25	100	4	16	0.75	3	10	40	
TOTAL:		16		700.0		85.0		16.5		112.0	
				43.8		5.3		1.0		7.0	1.2
UNDERCUT											
BQFG	A5/R2	3	75	225	9	27	1.5	4.5	3	9	
PEGMATOID	A5/R2	9.5	60	570	7.5	71.25	1.5	14.25	4	38	
	A7/R2	3.5	50	175	6	21	1.5	5.3	6	21	
TOTAL:		16		970.0		119.3		24.0		68.0	
				60.6		7.5		1.5		4.3	2.9
									AVERAGE:		2.0

DDH's	RQD in HW:
3140	72
3139	97
3446	63
3167	78
3145.0	77.0
3146.0	87.0

Q' Weighted	RQD	Jn	Jr	Ja
2.5	72.3	6.4	1.3	5.6

A	B	C	N'
1	0.2	4.1	2.1

170-870

RMR = 9 ln Q' + 44

OVERCUT		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
PEGMATOID	A5/R2	6	63	378	2.5	15
	A5/R1	6	25	150	0.3	1.8
	A7/R1	4	38	152	0.3	1.2
TOTAL:		16		680.0		18.0
				42.5		1.1

UNDERCUT						
BQFG	A5/R2	3	68	204	8.3	24.9
PEGMATOID	A5/R2	9.5	63	598.5	2.5	23.75
	A7/R2	3.5	73	255.5	4.7	16.5
TOTAL:		16		1058.0		65.1
				66.1		4.1

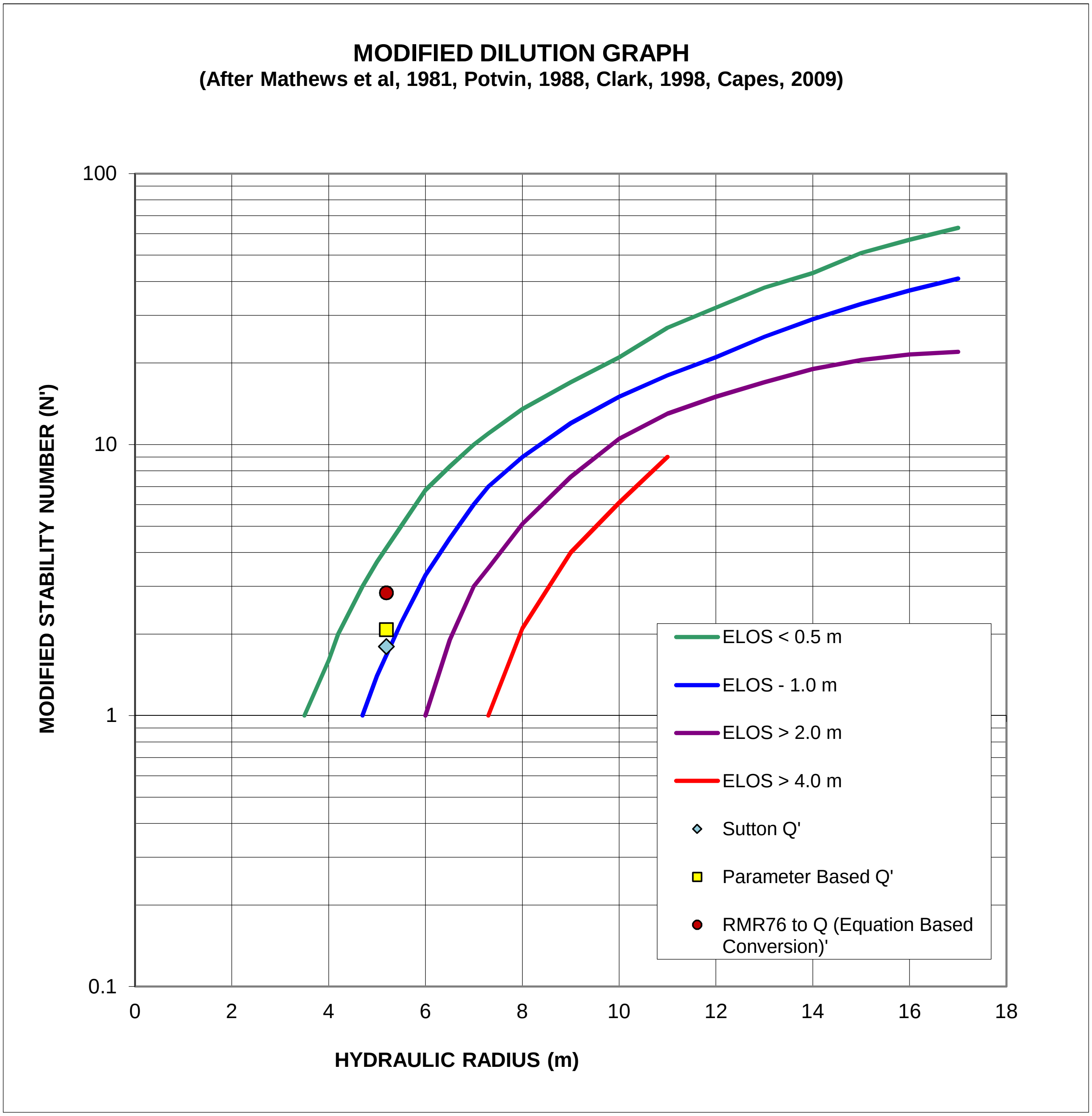
DDH's	RQD in HW:
3140	72
3139	97
3446	63
3167	78
3145.0	77.0
3146.0	87.0

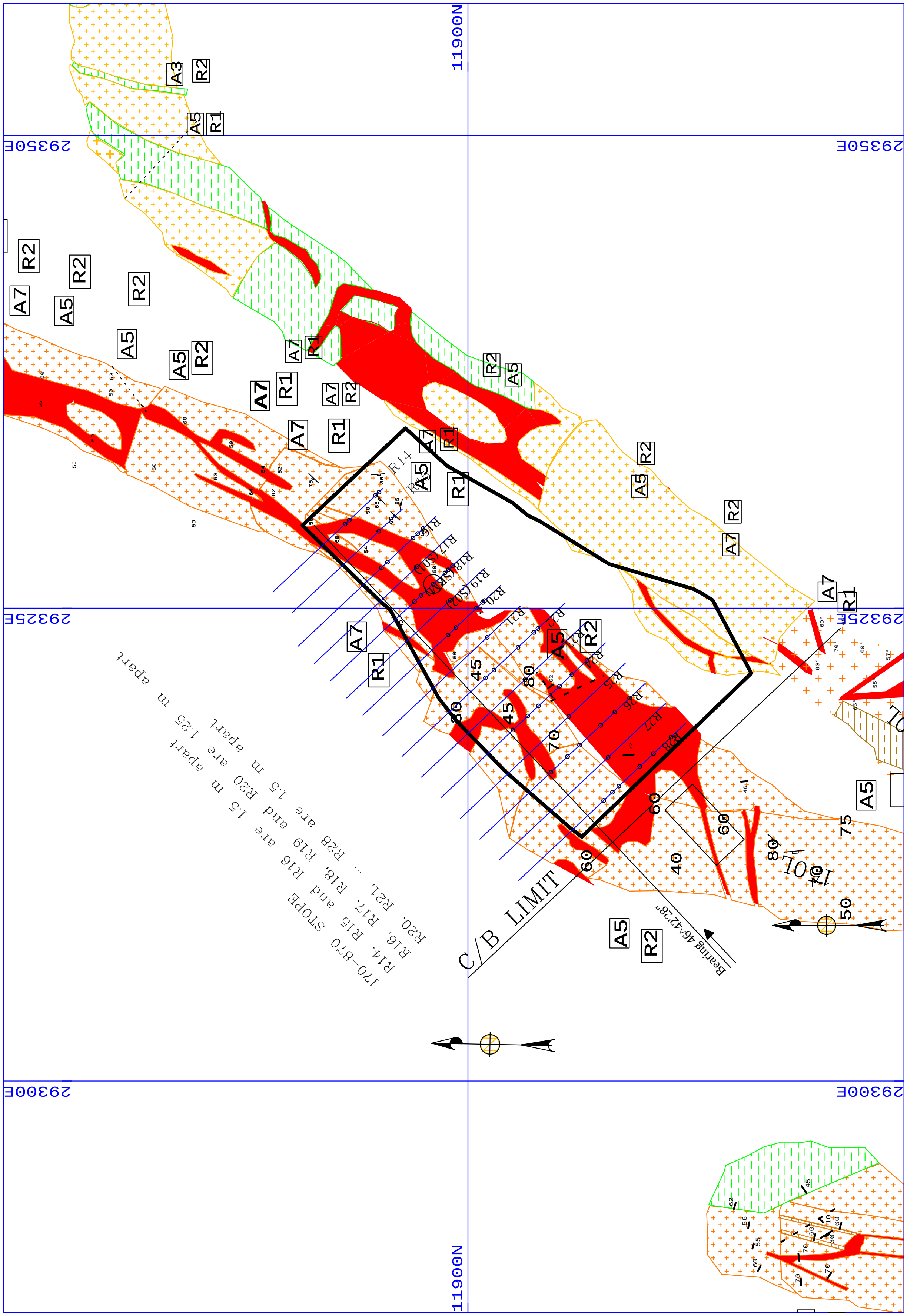
Q' Weighted	RQD	RQD O/C and U/C	Q'
3.5	72.8	54.3	2.6

A	B	C	N'
1	0.2	4.1	2.8

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	170-870		
Modified Stability Number (N')	Sutton N'	1.8	
	Parameter Based N'	2.1	
	RMR76 to Q (Equation Based Conversion)	2.8	
Hydraulic Radius	Actual	5.2	
Average HW Overbreak from the Optech:		**3.1**	**m**




11900N
29350E
29325E
29300E
C/B LIMIT

STOPE	170-900	
Strike Length	23.5	
HW Exposed Height	29	
Hydraulic Radius	6.5	
Average HW Dip	50.6	°
Percentage of HW that is:		
Gneissic	35	%
Pegmatoidal	65	%
Other	0	%

170-900 — Sutton's Q' from A/R values

OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
Pegmatoid	A7/R2	3.5	60	210	6	21	1.5	5.25	6	21	
	A7/R1	9	20	180	4	36	0.75	6.75	10	90	
	A5/R2	11	75	825	7.5	82.5	1.5	16.5	6	66	
TOTAL:		23.5		1215.0		139.5		28.5		177.0	
				51.7		5.9		1.2		7.5	1.4
UNDERCUT											
BQFG	A5/R2	2.5	75	187.5	7.5	18.75	1.5	3.75	6	15	
	A7/R2	4	60	240	6	24	1.5	6	6	24	
	A7/R1	6.5	20	130.0	4	26.0	0.8	4.9	10.0	65.0	
	A5/R1	3.5	50	175.0	4.0	14.0	0.8	2.6	8.0	28.0	
Pegmatoid	A5/R1	3.5	50	175.0	4.0	14.0	0.75	2.6	8.0	28.0	
	A3/R2	3.5	90	315.0	9.0	31.5	1.5	5.3	4.0	14.0	
TOTAL:		23.5		1222.5		128.3		25.1		174.0	
				52.0		5.5		1.1		7.4	1.4

DDH's	RQD in HW:
3139	69
3448	96
3135.0	97.0
3136.0	100.0
3449.0	72.0
3300	95
3439.0	93.0
3291	100
3295	90
3292	90

Q' Weighted	RQD	Jn	Jr	Ja
2.2	83.8	5.7	1.1	7.5

A	B	C	N'
1	0.2	4.2	1.9

170-900

Parameter Based Q' from A/R values

		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
OVERCUT											
Pegmatoid	A7/R2	3.5	50	175	6	21	1.5	5.25	6	21	
	A7/R1	9	25	225	4	36	0.75	6.75	10	90	
	A5/R2	11	60	660	7.5	82.5	1.5	16.5	4	44	
TOTAL:		23.5		1060.0		139.5		28.5		155.0	
				45.1		5.9		1.2		6.6	1.4
UNDERCUT											
BQFG	A5/R2	2.5	75	187.5	9	22.5	1.5	3.75	3	7.5	
	A7/R2	4	75	300	9	36	1.5	6	4	16	
	A7/R1	6.5	75	487.5	9	58.5	0.75	4.9	4	26	
	A5/R1	3.5	75	262.5	9.0	31.5	0.8	2.6	4.0	14.0	
Pegmatoid	A5/R1	3.5	40	140.0	4.0	14.0	0.75	2.6	8.0	28.0	
	A3/R2	3.5	75	262.5	9.0	31.5	1.5	5.3	3.0	10.5	
TOTAL:		23.5		1640.0		194.0		25.1		102.0	
				69.8		8.3		1.1		4.3	2.1

DDH's	RQD in HW:
3139	69
3448	96
3135.0	97.0
3136.0	100.0
3449.0	72.0
3300	95
3439.0	93.0
3291	100
3295	90
3292	90

Q' Weighted	RQD	Jn	Jr	Ja
2.5	84.7	7.1	1.1	5.5

A	B	C	N'
1	0.2	4.2	2.1

170-900

RMR = 9 ln Q' + 44

		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
OVERCUT						
Pegmatoid	A7/R2	3.5	73	255.5	4.7	16.45
	A7/R1	9	38	342	0.3	2.7
	A5/R2	11	63	693	2.5	27.5
TOTAL:		23.5		1290.5		46.7
				54.9		2.0
UNDERCUT						
BQFG	A5/R2	2.5	68	170	8.3	20.75
	A7/R2	4	73	292	5	20
	A7/R1	6.5	69	448.5	3.6	23.4
	A5/R1	3.5	38	133.0	0.7	2.5
Pegmatoid	A5/R1	3.5	25	87.5	0.3	1.1
	A3/R2	3.5	71	248.5	5.9	20.7
TOTAL:		23.5		1379.5		88.3
				58.70212766		3.757446809

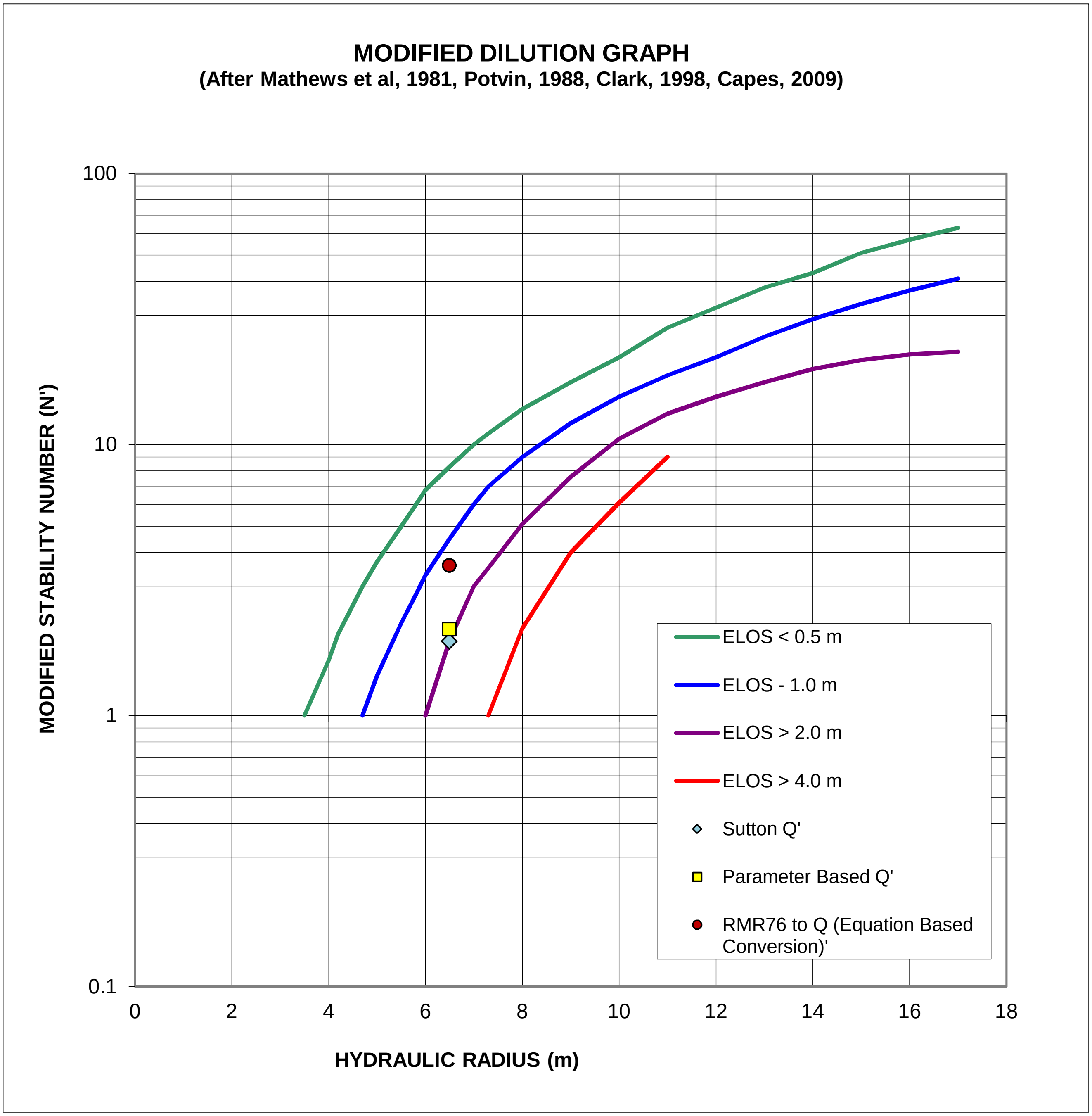
DDH's	RQD in HW:
3139	69
3448	96
3135.0	97.0
3136.0	100.0
3449.0	72.0
3300	95
3439.0	93.0
3291	100
3295	90
3292	90

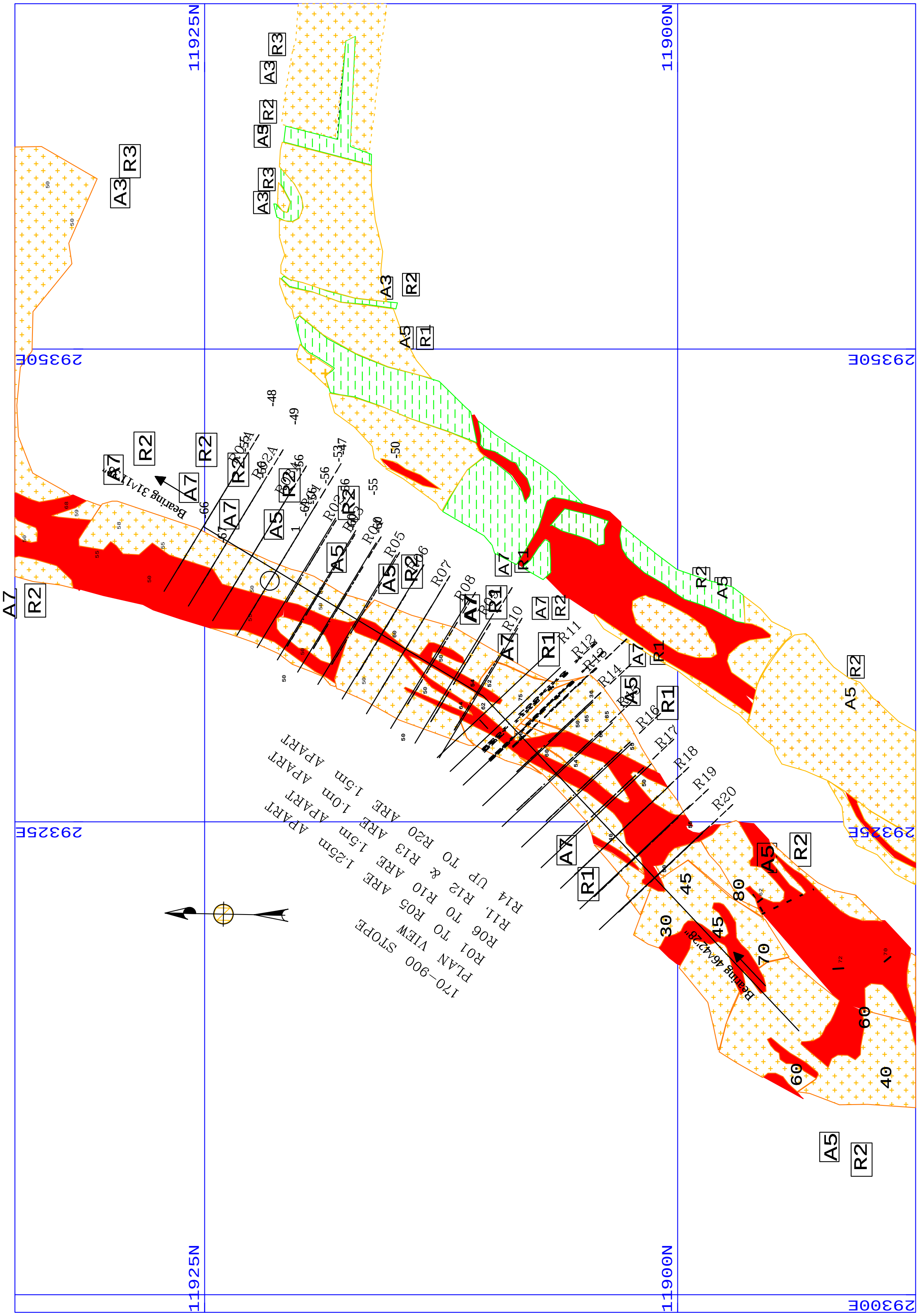
Q' Weighted	RQD	RQD O/C and U/C	Q'
4.3	84.6	56.8	2.9

A	B	C	N'
1	0.2	4.2	3.6

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	170-900	
Modified Stability Number (N')	Sutton N'	1.9
	Parameter Based N'	2.1
	RMR76 to Q (Equation Based Conversion)	3.6
Hydraulic Radius	Actual	6.5
Average HW Overbreak from the Optech:		**1.1** **m**




PLAN VIEW STOPE 170-900
R01 TO R05 ARE 1.25m APART
R06 TO R10 ARE 1.5m APART
R11, R12 & R13 ARE 1.0m APART
R14 UP TO R20 ARE 1.5m APART
11925N
11900N
29350E
29325E
29300E
Bearing 311
Bearing 46

STOPE	245-055	
Strike Length	21	
HW Exposed Height	20.5	
Hydraulic Radius	5.2	
Average HW Dip	35.7	°
Percentage of HW that is:		
Gneissic	98	%
Pegmatoidal	2	%
Other	0	%

Sutton's Q' from A/R values

245-055											
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
BQFG	A5/R1	14	50	700	4	56	0.75	10.5	8	112	
	A5/R3	4.8	100	480	9	43.2	1.5	7.2	2	9.6	
PEGMATOID	A7/R1	2.2	20	44	4	8.8	0.75	1.65	10	22	
TOTAL:		21		1224.0		108.0		19.4		143.6	
				58.3		5.1		0.9		6.8	1.5
UNDERCUT											
BQFG	A5/R2	5	75	375	7.5	37.5	1.5	7.5	6	30	
	A7/R1	16	20	320	4	64	0.75	12	10	160	
TOTAL:		21		695.0		101.5		19.5		190.0	
				33.1		4.8		0.9		9.0	0.7
									AVERAGE:		1.1

DDH's RQD in HW:
None logged

Q' Weighted	RQD	Jn	Jr	Ja
1	46	5.0	0.9	7.9

A	B	C	N'
1.0	0.2	3.1	0.7

245-055

Parameter Based Q' from A/R values

OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
BQFG	A5/R1	14	75	1050	9	126	0.75	10.5	4	56	
	A5/R3	4.8	75	360	9	43.2	1.5	7.2	2	9.6	
PEGMATOID	A7/R1	2.2	25	55	4	8.8	0.75	1.65	10	22	
TOTAL:		21		1465.0		178.0		19.4		87.6	
				69.8		8.5		0.9		4.2	1.8
UNDERCUT											
BQFG	A5/R2	5	75	375	9	45	1.5	7.5	3	15	
	A7/R1	16	75	1200	9	144	0.75	12	4	64	
TOTAL:		21		1575		189		19.5		79	
				75.0		9.0		0.9		3.8	2.1
									AVERAGE:		1.9

DDH's RQD in HW:

None logged

Q' Weighted	RQD	Jn	Jr	Ja
2	72	8.7	0.9	4.0

A	B	C	N'
1.0	0.2	3.1	1.2

245-055

RMR = 9 ln Q' + 44

		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
OVERCUT						
BQFG	A5/R1	14	38	532	0.7	9.8
	A5/R3	4.8	73	350.4	14	67.2
PEGMATOID	A7/R1	2.2	38	83.6	0.3	0.66
TOTAL:		21		966.0		77.7
				46.0		3.7
UNDERCUT						
BQFG	A5/R2	5	68	340	8.3	41.5
	A7/R1	16	69	1104	3.6	57.6
TOTAL:		21		1444.0		99.1
				68.8		4.7

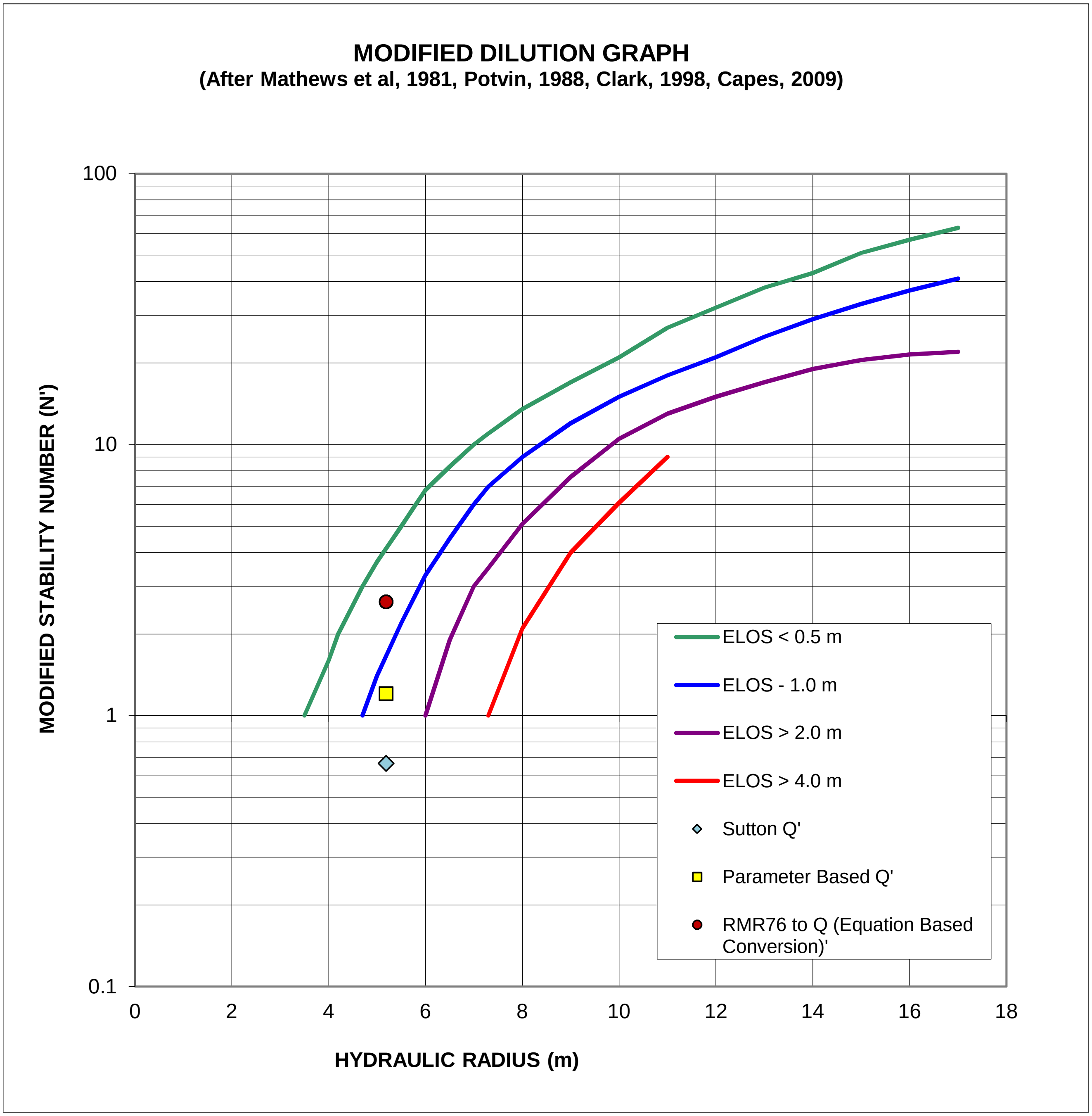
DDH's RQD in HW:
None logged

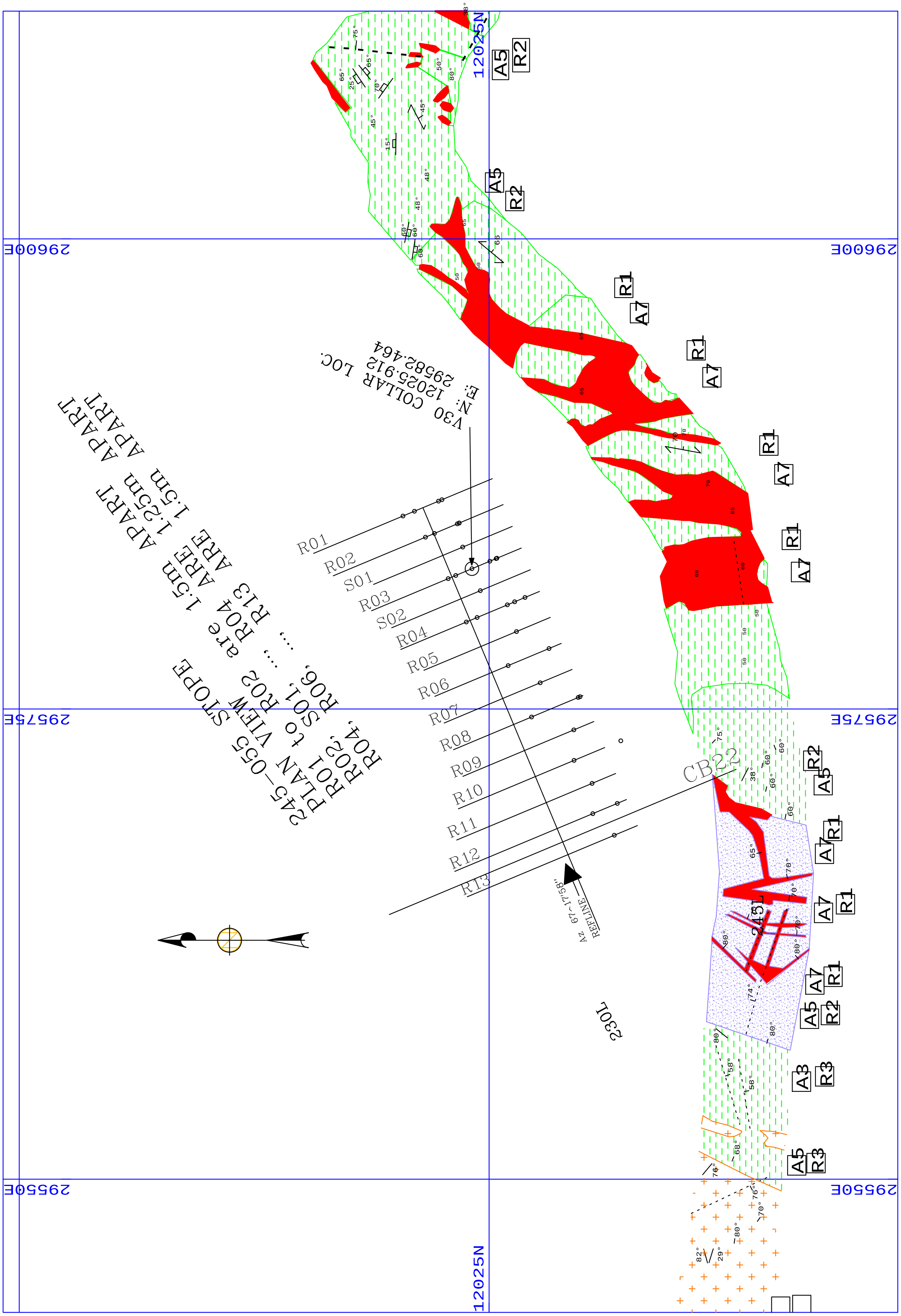
Q' Weighted	RQD	RQD O/C and U/C	Q'
4	57	57.4	4.2

A	B	C	N'
1.0	0.2	3.1	2.6

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	245-055	
Modified Stability Number (N')	Sutton N'	0.7
	Parameter Based N'	1.2
	RMR76 to Q (Equation Based Conversion)	2.6
Hydraulic Radius	Actual	5.2
Average HW Overbreak from the Optech:		**2.7** m




12025N
29600E
29575E
29550E
V30 COLLAR LOC.
N: 12025.912
E: 29582.464
R01
R02
S01
R03
S02
R04
R05
R06
R07
R08
R09
R10
R11
R12
R13
CB22
245L
230L
A5
R2
R1
A7
A3
R3

STOPE	245-065	
Strike Length	21.7	
HW Exposed Height	21.9	
Hydraulic Radius	5.4	
Average HW Dip	48.7	°
Percentage of HW that is:		
Gneissic	100	%
Pegmatoidal	0	%
Other	0	%

Sutton's Q' from A/R values

245-065											
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
BQFG	A5/R2	16.5	75	1237.5	7.5	123.75	1.5	24.75	6	99	
	A5/R3	5.2	100	520	9	46.8	1.5	7.8	2	10.4	
TOTAL:		21.7		1757.5		170.55		32.55		109.4	
				81.0		7.9		1.5		5.0	3.1
UNDERCUT											
BQFG	A5/R2	13.7	75	1027.5	7.5	102.8	1.5	20.6	6.0	82.2	
	A7/R1	8	20	160	4	32	0.75	6	10	80	
TOTAL:		21.7		1187.5		134.75		26.55		162.2	
				54.7		6.2		1.2		7.5	1.4
									AVERAGE:		2.3

DDH's	RQD in HW:
1721	27

Q' Weighted	RQD	Jn	Jr	Ja
1.7	54.2	7.0	1.4	6.3

A	B	C	N'
1.0	0.2	4.0	1.4

245-065

Parameter Based Q' from A/R values

		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
OVERCUT											
BQFG	A5/R2	16.5	75	1237.5	9	148.5	1.5	24.75	3	49.5	
	A5/R3	5.2	75	390	9	46.8	1.5	7.8	2	10.4	
TOTAL:		21.7		1627.5		195.3		32.55		59.9	
				75.0		9.0		1.5		2.8	4.5
UNDERCUT											
BQFG	A5/R2	13.7	75	1027.5	9.0	123.3	1.5	20.6	3.0	41.1	
	A7/R1	8	75	600	9	72	0.75	6	4	32	
TOTAL:		21.7		1627.5		195.3		26.55		73.1	
				75		9		1.2		3.368663594	3.0
									AVERAGE:		3.8

DDH's	RQD in HW:
1721	27

Q' Weighted	RQD	Jn	Jr	Ja
2.9	59.0	9.0	1.4	3.1

A	B	C	N'
1.0	0.2	4.0	2.4

245-065

RMR = 9 ln Q' + 44

		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
OVERCUT						
BQFG	A5/R2	16.5	68	1122	8.3	136.95
	A5/R3	5.2	73	379.6	14	72.8
TOTAL:		21.7		1501.6		209.75
				69.2		9.7
UNDERCUT						
BQFG	A5/R2	13.7	68	931.6	8.3	113.7
	A7/R1	8	69	552	3.6	28.8
TOTAL:		21.7		1483.6		142.51
				68.4		6.6

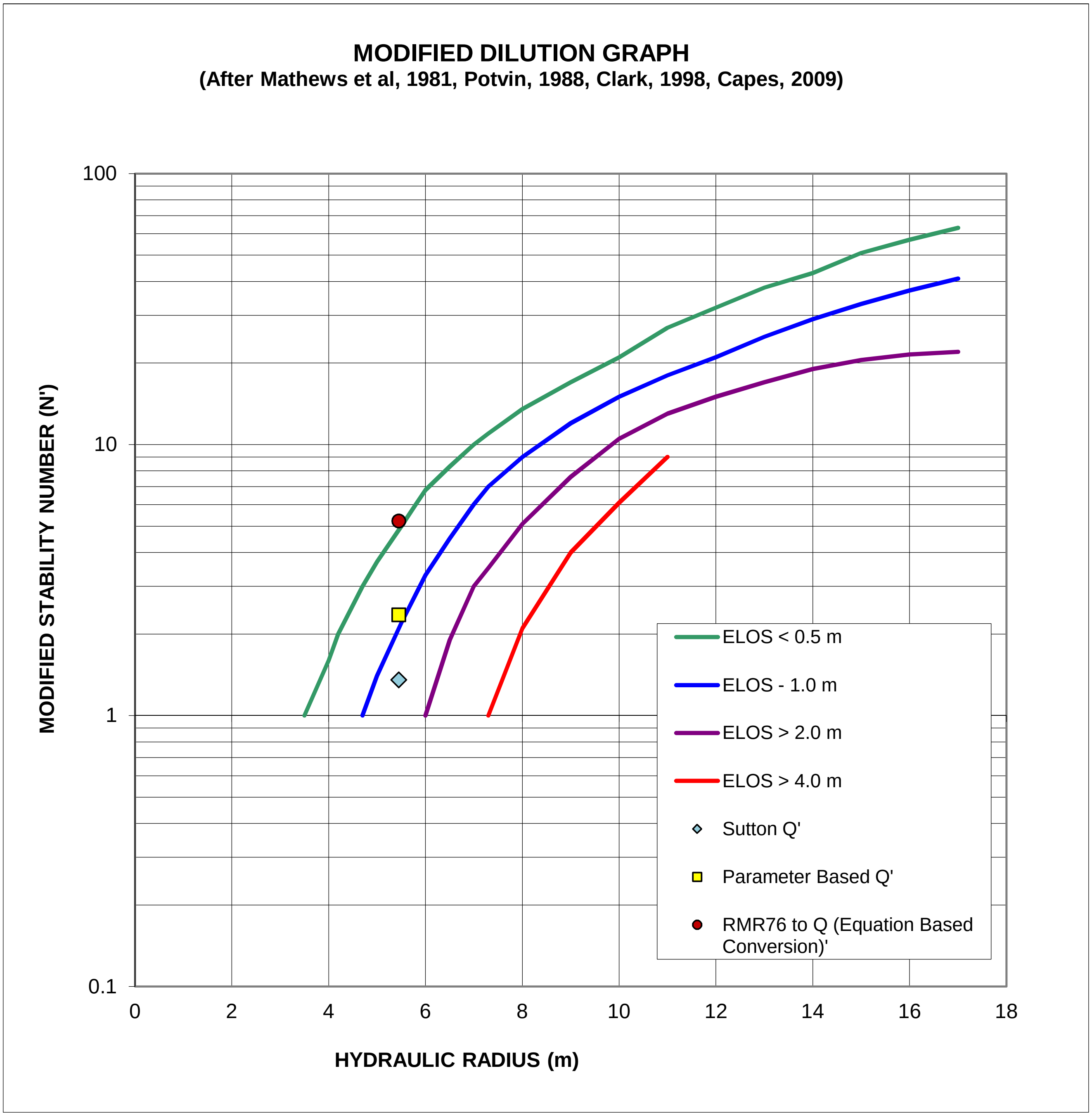
DDH's	RQD in HW:
1721	27

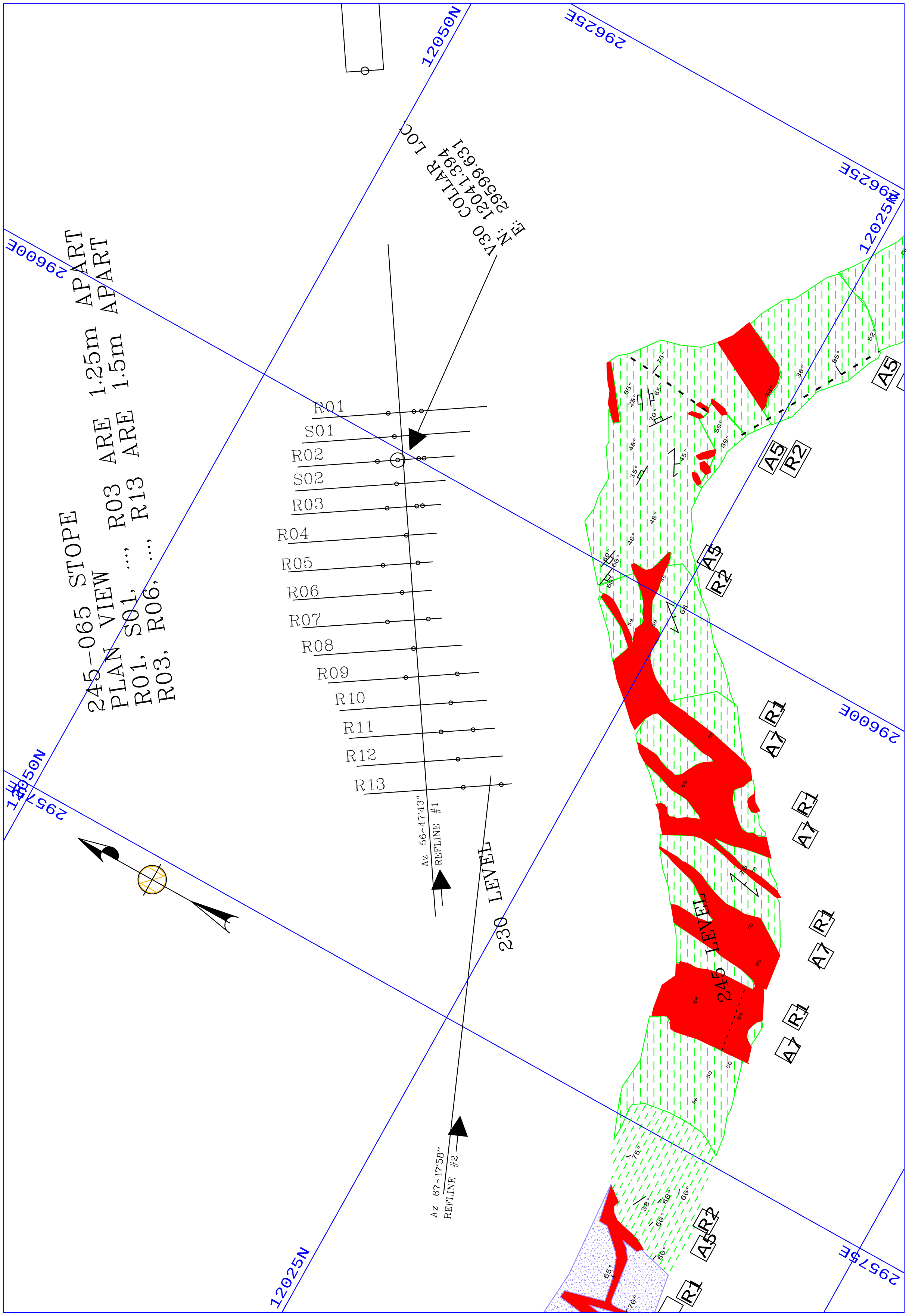
Q' Weighted	RQD	RQD O/C and U/C	Q'
6.5	54.9	68.8	8.1

A	B	C	N'
1.0	0.2	4.0	5.2

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	245-065	
Modified Stability Number (N')	Sutton N'	1.4
	Parameter Based N'	2.4
	RMR76 to Q (Equation Based Conversion)	5.2
Hydraulic Radius	Actual	5.4
Average HW Overbreak from the Optech:		**0.8** **m**




245-065 STOPE
PLAN VIEW
R01, S01, ..., R03 ARE 1.25m APART
R03, R06, ..., R13 ARE 1.5m APART
V30 COLLAR LOC
N: 12041.394
E: 29599.631
12050N
12025N
29625E
29600E
29575E
R01
S01
R02
S02
R03
R04
R05
R06
R07
R08
R09
R10
R11
R12
R13
Az 56~47'43''
REFLINE #1
Az 67~17'58''
REFLINE #2
230 LEVEL
245 LEVEL
A5
R2
A7
R1

STOPE	252-1	
Strike Length	37	
HW Exposed Height	24.9	
Hydraulic Radius	7.4	
Average HW Dip	55.1	°
Percentage of HW that is:		
Gneissic	90	%
Pegmatoidal	0	%
Other	10	% (Feldspar Porphyry)

Sutton's Q' from A/R values

252-1											
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
BQFG	A7/R1	17.5	20	350	4	70	0.75	13.125	10	175	
	A5/R3	4	100	400	9	36	1.5	6	2	8	
	A7/R2	4	60	240	6	24	1.5	6	6	24	
	A5/R1	4	50	200.0	4	16.0	0.8	3.0	8.0	32.0	
FXPO	A5/R2	5.5	75	412.5	7.5	41.3	1.5	8.3	6.0	33.0	
	A7/R1	2	20	40.0	4.0	8.0	0.8	1.5	10.0	20.0	
TOTAL:		37		1642.5		195.3		37.9		292.0	
				44.4		5.3		1.0		7.9	1.1
UNDERCUT											
BQFG	A5/R2	30.5	75	2287.5	7.5	228.8	1.5	45.8	6.0	183.0	
	A7/R2	6.5	60	390.0	6.0	39.0	1.5	9.8	6.0	39.0	
TOTAL:		37		2677.5		267.8		55.5		222.0	
				72.4		7.2		1.5		6.0	2.5

DDH's	RQD in HW:
2274.0	63.0
2282	75
2616	83
2615.0	69.0
2288.0	90.0

Q' Weighted	RQD	Jn	Jr	Ja
2.1	71.0	6.3	1.3	6.9

A	B	C	N'
1	0.2	4.6	1.9

Parameter Based Q' from A/R values

252-1											
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
BQFG	A7/R1	17.5	75	1312.5	9	157.5	0.75	13.125	4	70	
	A5/R3	4	75	300	9	36	1.5	6	2	8	
	A7/R2	4	75	300	9	36	1.5	6	4	16	
	A5/R1	4	75	300.0	9.0	36.0	0.75	3.0	4	16.0	
FXPO	A5/R2	5.5	60	330.0	7.5	41.3	1.5	8.3	4.0	22.0	
	A7/R1	2	25	50	4	8.0	0.8	1.5	10.0	20.0	
TOTAL:		37		2592.5		314.8		37.9		152.0	
				70.1		8.5		1.0		4.1	2.1
UNDERCUT											
BQFG	A5/R2	30.5	75	2287.5	9.0	274.5	1.5	45.8	3.0	91.5	
	A7/R2	6.5	75	487.5	9.0	58.5	1.5	9.8	4.0	26.0	
TOTAL:		37		2775.0		333.0		55.5		117.5	
				75.0		9.0		1.5		3.2	3.9

DDH's	RQD in HW:
2274.0	63.0
2282	75
2616	83
2615.0	69.0
2288.0	90.0

Q' Weighted	RQD	Jn	Jr	Ja
3.0	75.0	8.8	1.3	3.6

A	B	C	N'
1	0.2	4.6	2.7

252-1

$RMR = 9 \ln Q' + 44$

		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
OVERCUT						
BQFG	A7/R1	17.5	69	1207.5	3.6	63
	A5/R3	4	73	292	14	56
	A7/R2	4	73	292	5	20
	A5/R1	4	38	152.0	0.7	2.8
FXPO	A5/R2	5.5	63	346.5	2.5	13.8
	A7/R1	2	38	76.0	0.3	0.6
TOTAL:		37		2366.0		156.2
				63.9		4.2
UNDERCUT						
BQFG	A5/R2	30.5	68	2074.0	8.3	253.2
	A7/R2	6.5	73	474.5	5.0	32.5
TOTAL:		37		2548.5		285.7
				68.9		7.7

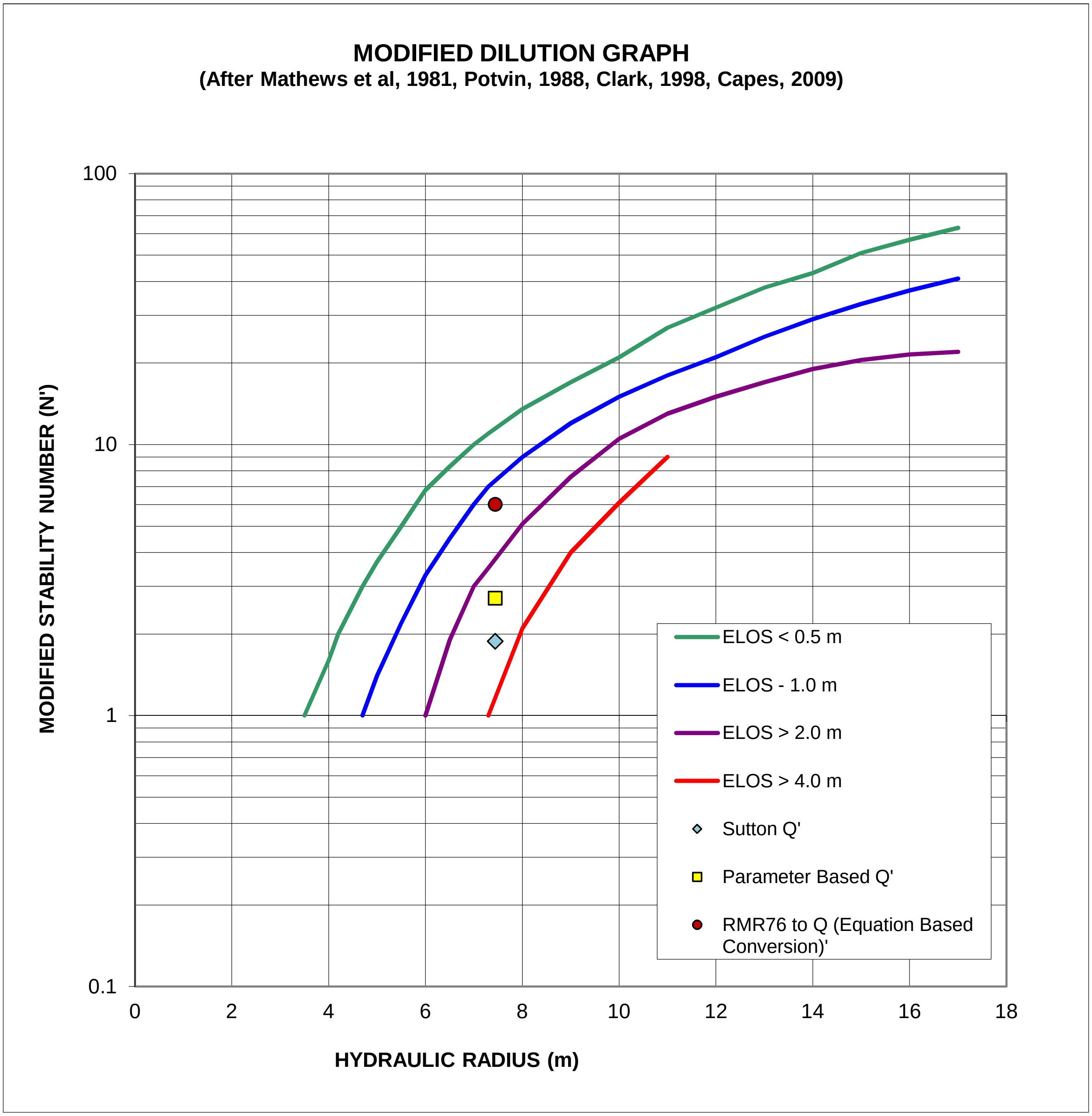
DDH's	RQD in HW:
2274.0	63.0
2282	75
2616	83
2615.0	69.0
2288.0	90.0

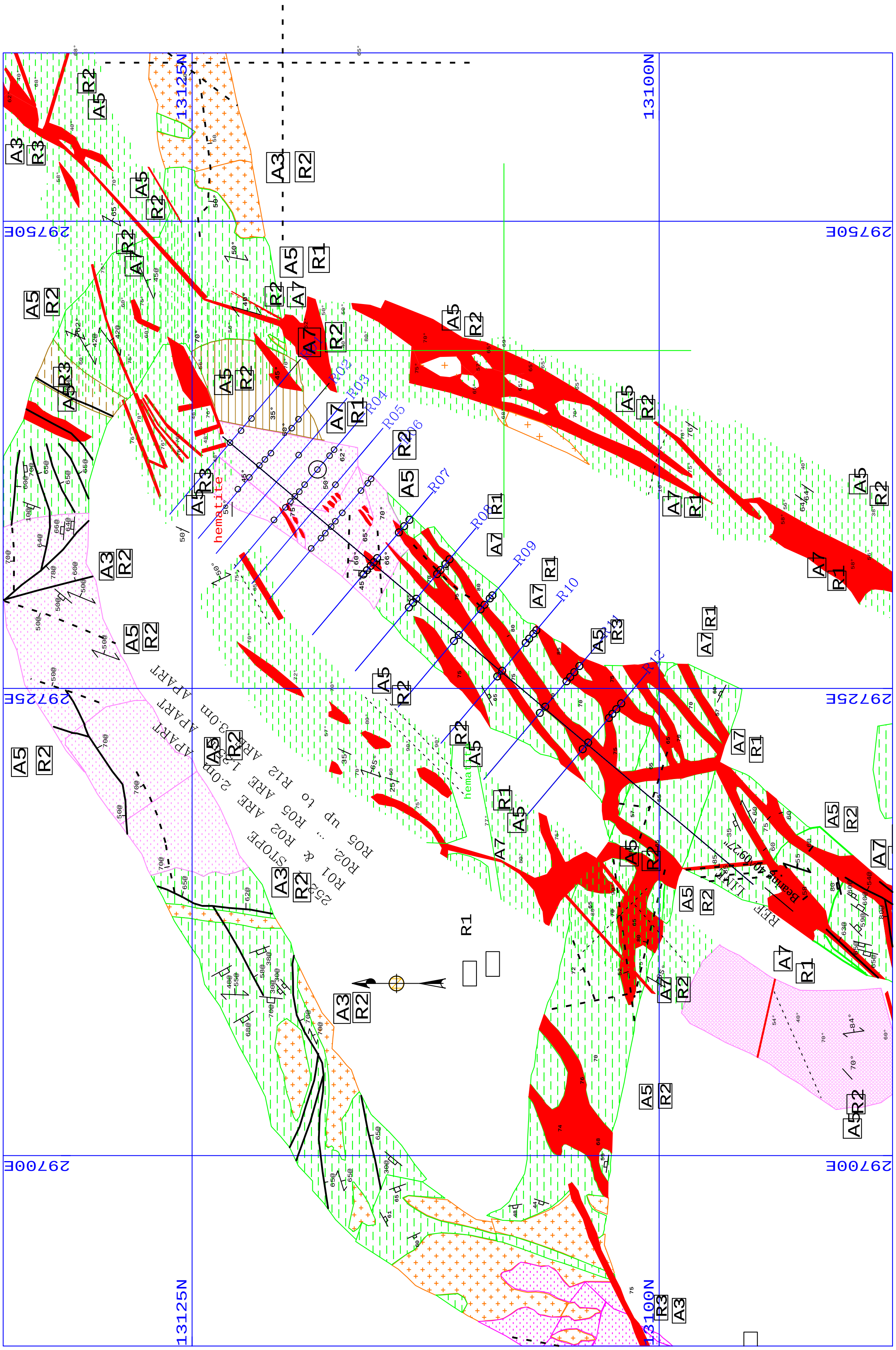
Q' Weighted	RQD	RQD O/C and U/C	Q'
6.6	73.3	66.4	6.0

A	B	C	N'
1	0.2	4.6	6.0

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	252-1	
Modified Stability Number (N')	Sutton N'	1.9
	Parameter Based N'	2.7
	RMR76 to Q (Equation Based Conversion)	6.0
Hydraulic Radius	Actual	7.4
Average HW Overbreak from the Optech:		**1.6** m




13125N
13100N
hematite
R02
R03
R04
R05
R06
R07
R08
R09
R10
R12
A3
A5
A7
R1
R2
R3

STOPE	252-3	
Strike Length	22.5	
HW Exposed Height	25.6	
Hydraulic Radius	6.0	
Average HW Dip	51.7	°
Percentage of HW that is:		
Gneissic	92	%
Pegmatoidal	8	%
Other	0	%

Sutton's Q' from A/R values

252-3												
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
BQFG	A5/R2	15.5	75	1162.5	7.5	116.25	1.5	23.25	6	93	
	A7/R1	7	20	140	4	28	0.75	5.25	10	70	
TOTAL:		22.5		1302.5		144.25		28.5		163	
				57.9		6.4		1.3		7.2	1.6
UNDERCUT											
Pegmatoid	A5/R2	3.5	75	262.5	7.5	26.3	1.5	5.3	6.0	21.0	
BQFG	A5/R2	9	75	675.0	7.5	67.5	1.5	13.5	6.0	54.0	
	A5/R3	4	100	400	9	36	1.5	6	2	8	
	A7/R1	6	20	120.0	4	24.0	0.8	4.5	10.0	60.0	
TOTAL:		22.5		1457.5		153.8		29.3		143.0	
				64.8		6.8		1.3		6.4	1.9

DDH's	RQD in HW:
2619	70

Q' Weighted	RQD	Jn	Jr	Ja
2	64	6.6	1.3	6.8

A	B	C	N'
1.0	0.2	4.3	1.6

252-3

Parameter Based Q' from A/R values

OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
BQFG	A5/R2	15.5	75	1162.5	9	139.5	1.5	23.25	3	46.5	
	A7/R1	7	75	525	9	63	0.75	5.25	4	28	
TOTAL:		22.5		1687.5		202.5		28.5		74.5	
				75.0		9.0		1.3		3.3	3.2
UNDERCUT											
Pegmatoid	A5/R2	3.5	60	210.0	7.5	26.3	1.5	5.3	4.0	14.0	
BQFG	A5/R2	9	75	675.0	9.0	81.0	1.5	13.5	3.0	27.0	
	A5/R3	4	75	300	9	36	1.5	6	2	8	
	A7/R1	6	75	450	9	54	0.75	4.5	4	24	
TOTAL:		22.5		1635.0		197.3		29.3		73.0	
				72.7		8.8		1.3		3.2	3.3

DDH's	RQD in HW:
2619	70

Q' Weighted	RQD	Jn	Jr	Ja
3	73	8.9	1.3	3.3

A	B	C	N'
1.0	0.2	4.3	2.7

252-3			RMR = 9 ln Q' + 44			
OVERCUT		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
BQFG	A5/R2	15.5	68	1054	8.3	128.65
	A7/R1	7	69	483	3.6	25.2
TOTAL:		22.5		1537		153.85
				68.3		6.8
UNDERCUT						
Pegmatoid	A5/R2	3.5	63	220.5	2.5	8.8
BQFG	A5/R2	9	68	612.0	8.3	74.7
	A5/R3	4	73	292	14	56
	A7/R1	6	69	414.0	3.6	21.6
TOTAL:		22.5		1538.5		161.1
				68.4		7.2

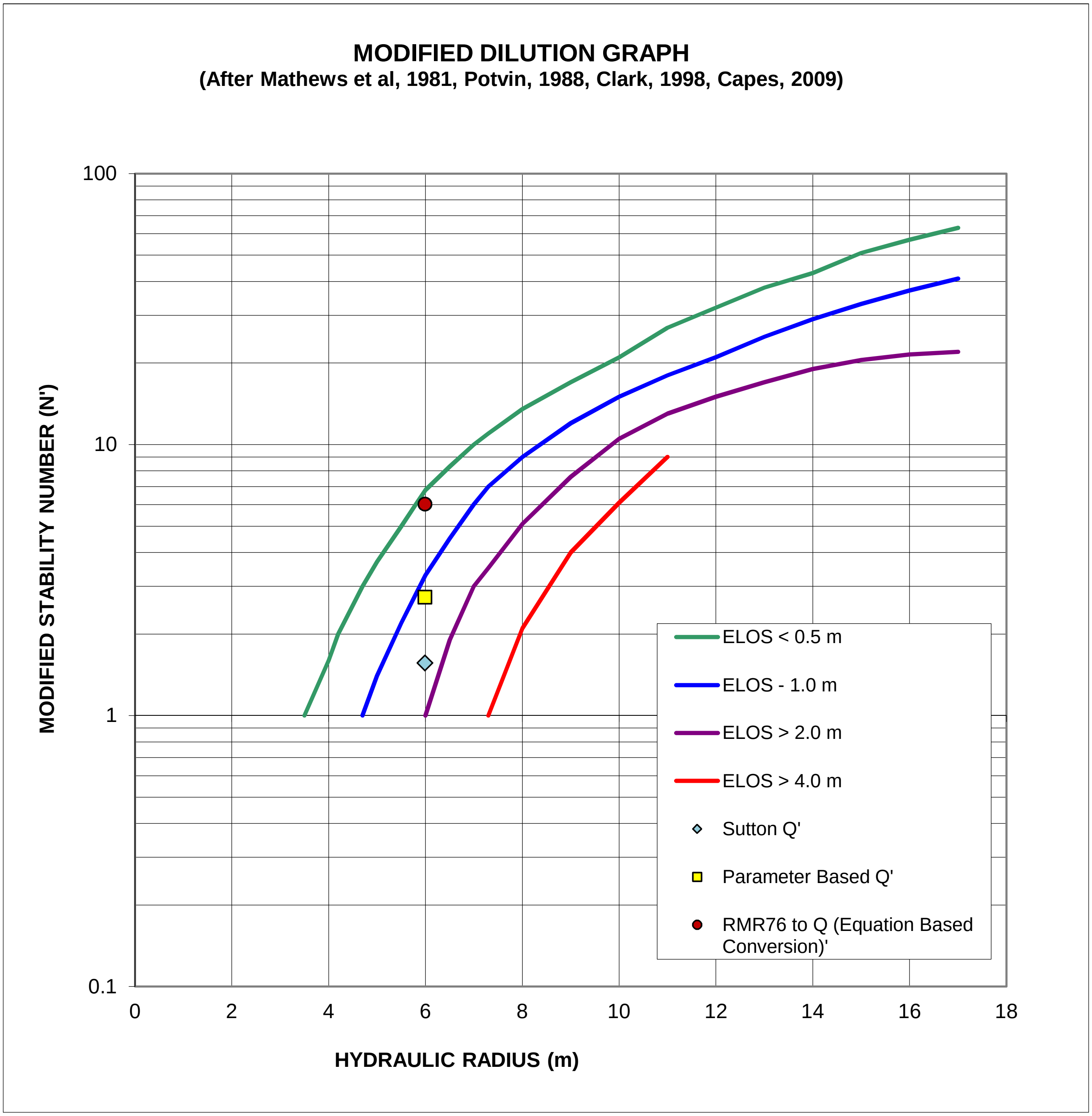
DDH's	RQD in HW:
2619	70

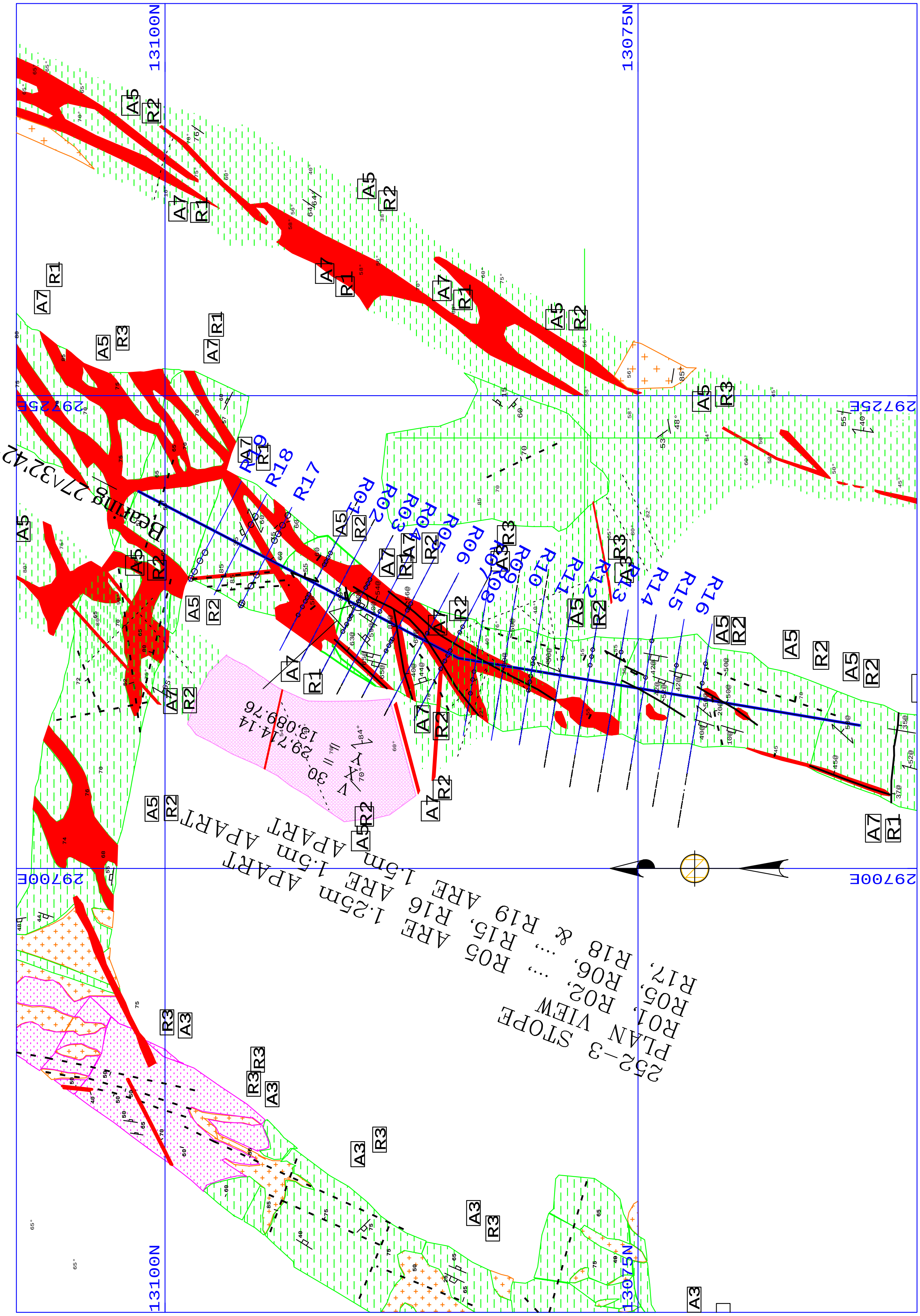
Q' Weighted	RQD	RQD O/C and U/C	Q'
7	69	68.3	7.0

A	B	C	N'
1.0	0.2	4.3	6.0

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	252-3	
Modified Stability Number (N')	Sutton N'	1.6
	Parameter Based N'	2.7
	RMR76 to Q (Equation Based Conversion)	6.0
Hydraulic Radius	Actual	6.0
Average HW Overbreak from the Optech:		**2.9** m




252-3 STOPE
PLAN VIEW
R01, R02, ... R05 ARE 1.25m APART
R05, R06, ... R15, R16 ARE 1.5m APART
R17, R18 & R19 ARE 1.5m APART
Bearing 27°32'42"
13100N
13075N
29725E
29700E

STOPE	252-4	
Strike Length	27.5	
HW Exposed Height	24	
Hydraulic Radius	6.4	
Average HW Dip	53.6	°
Percentage of HW that is:		
Gneissic	75	%
Pegmatoidal	25	%
Other	0	%

252-4			Sutton's Q' from A/R values								
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
BQFG	A5/R2	16.5	75	1237.5	7.5	123.75	1.5	24.75	6	99	
	A7/R1	11	20	220	4	44	0.75	8.25	10	110	
TOTAL:		27.5		1457.5		167.75		33		209	
				53.0		6.1		1.2		7.6	1.4
UNDERCUT											
BQFG	A5/R3	5	100	500.0	9.0	45.0	1.5	7.5	2.0	10.0	
	A5/R2	4	75	300.0	7.5	30.0	1.5	6.0	6.0	24.0	
	A5/R1	5	50	250	4	20	0.75	3.75	8	40	
Pegmatoid	A7/R1	11.5	20	230.0	4	46.0	0.8	8.6	10.0	115.0	
	A5/R2	2	75	150.0	7.5	15.0	1.5	3.0	6.0	12.0	
TOTAL:		27.5		1430.0		156.0		28.9		201.0	
				52.0		5.7		1.1		7.3	1.3

DDH's	RQD in HW:
2272	20
2622.0	30.0
2624	19

Q' Weighted	RQD	Jn	Jr	Ja
0.9	34.8	5.9	1.1	7.5

A	B	C	N'
1.0	0.2	4.4	0.8

252-4

Parameter Based Q' from A/R values

OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
BQFG	A5/R2	16.5	75	1237.5	9	148.5	1.5	24.75	3	49.5	
	A7/R1	11	75	825	9	99	0.75	8.25	4	44	
TOTAL:		27.5		2062.5		247.5		33		93.5	
				75.0		9.0		1.2		3.4	2.9
UNDERCUT											
BQFG	A5/R3	5	75	375.0	9.0	45.0	1.5	7.5	2.0	10.0	
	A5/R2	4	75	300.0	9.0	36.0	1.5	6.0	3.0	12.0	
	A5/R1	5	75	375	9	45	0.75	3.75	4	20	
Pegmatoid	A7/R1	11.5	25	287.5	4	46	0.75	8.6	10	115	
	A5/R2	2	60	120.0	7.5	15.0	1.5	3.0	4.0	8.0	
TOTAL:		27.5		1457.5		187.0		28.9		165.0	
				53.0		6.8		1.1		6.0	1.4

DDH's	RQD in HW:
2272	20
2622.0	30.0
2624	19

Q' Weighted	RQD	Jn	Jr	Ja
1.2	39.4	7.9	1.1	4.7

A	B	C	N'
1.0	0.2	4.4	1.1

252-4

RMR = 9 ln Q' + 44

OVERCUT		Length (m)	RQD	RQD (weighted)	Q'	Q'(weighted)
BQFG	A5/R2	16.5	68	1122	8.3	136.95
	A7/R1	11	69	759	3.6	39.6
TOTAL:		27.5		1881		176.55
				68.4		6.4
UNDERCUT						
BQFG	A5/R3	5	73	365.0	14.0	70.0
	A5/R2	4	68	272.0	8.3	33.2
	A5/R1	5	38	190	0.7	3.5
Pegmatoid	A7/R1	11.5	38	437.0	0.3	3.5
	A5/R2	2	63	126.0	2.5	5.0
TOTAL:		27.5		1390.0		115.2
				50.5		4.2

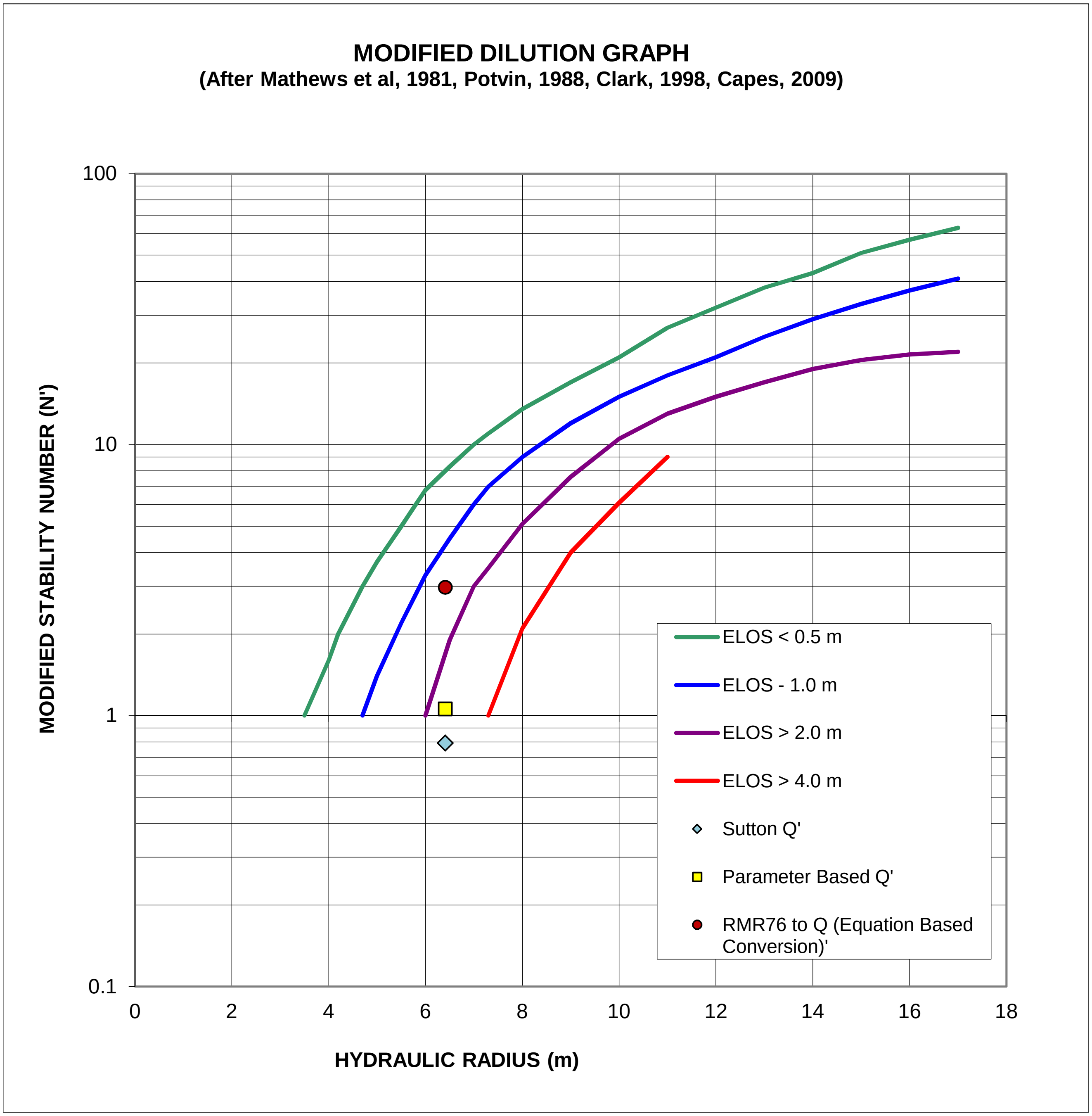
DDH's	RQD in HW:
2272	20
2622.0	30.0
2624	19

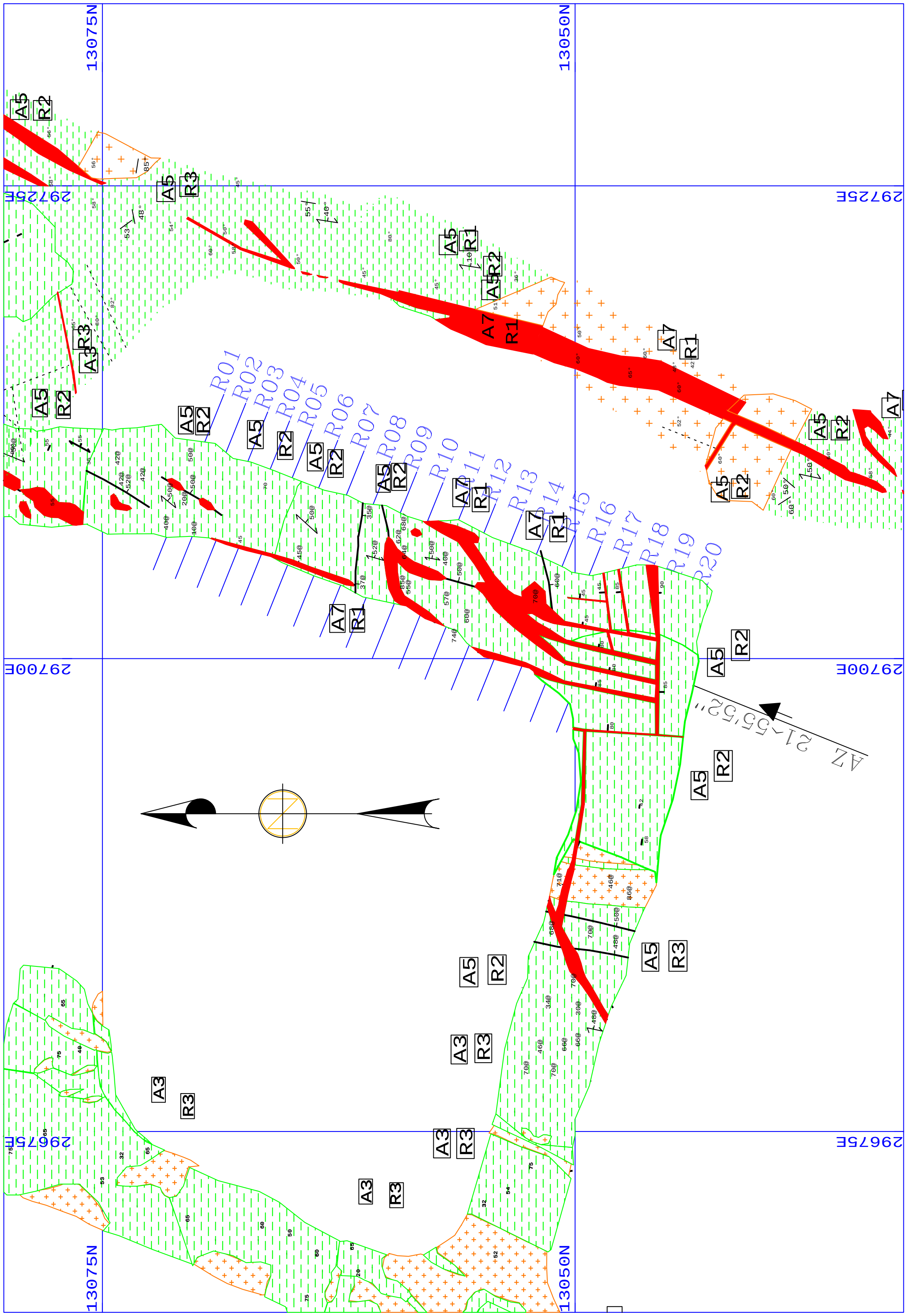
Q' Weighted	RQD	RQD O/C and U/C	Q'
3.4	37.6	59.5	5.3

A	B	C	N'
1.0	0.2	4.4	3.0

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	252-4	
Modified Stability Number (N')	Sutton N'	0.8
	Parameter Based N'	1.1
	RMR76 to Q (Equation Based Conversion)	3.0
Hydraulic Radius	Actual	6.4
Average HW Overbreak from the Optech:		**1.8** m




13075N
13050N
29725E
29700E
29675E
A5
R2
R3
A7
R1
A3
R01
R02
R03
R04
R05
R06
R07
R08
R09
R10
R11
R12
R13
R14
R15
R16
R17
R18
R19
R20
AZ 21-55'52"

STOPE	260-065	
Strike Length	30	
HW Exposed Height	24.4	
Hydraulic Radius	6.7	
Average HW Dip	44.2	°
Percentage of HW that is:		
Gneissic	94	%
Pegmatoidal	6	%
Other	0	%

Sutton's Q' from A/R values

260-065											
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
BQFG	A7/R1	19.5	20	390	4	78	0.75	14.625	10	195	
	A5/R2	10.5	75	787.5	7.5	78.75	1.5	15.75	6	63	
TOTAL:		30		1177.5		156.75		30.375		258	
				39.3		5.2		1.0		8.6	0.9
UNDERCUT											
BQFG	A7/R1	8	20	160.0	4.0	32.0	0.8	6.0	10.0	80.0	
	A5/R2	18.5	75	1387.5	7.5	138.8	1.5	27.8	6.0	111.0	
Pegmatoid	A3/R2	3.5	90	315	9	31.5	1.5	5.25	4	14	
TOTAL:		30		1862.5		202.3		39.0		205.0	
				62.1		6.7		1.3		6.8	1.8

DDH's	RQD in HW:
1723	63
2016	90
2017	70
3309.0	65.0

Q' Weighted	RQD	Jn	Jr	Ja
1.6	64.9	6.0	1.2	7.7

A	B	C	N'
1	0.2	3.7	1.2

260-065

Parameter Based Q' from A/R values

OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
BQFG	A7/R1	19.5	75	1462.5	9	175.5	0.75	14.625	4	78	
	A5/R2	10.5	75	787.5	9	94.5	1.5	15.75	3	31.5	
TOTAL:		30		2250		270		30.375		109.5	
				75.0		9.0		1.0		3.7	2.3
UNDERCUT											
BQFG	A7/R1	8	75	600.0	9.0	72.0	0.8	6.0	4.0	32.0	
	A5/R2	18.5	75	1387.5	9.0	166.5	1.5	27.8	3.0	55.5	
Pegmatoid	A3/R2	3.5	75	262.5	9	31.5	1.5	5.25	3	10.5	
TOTAL:		30		2250		270		39.0		98	
				75.0		9.0		1.3		3.3	3.3

DDH's	RQD in HW:
1723	63
2016	90
2017	70
3309.0	65.0

Q' Weighted	RQD	Jn	Jr	Ja
2.7	73.0	9.0	1.2	3.5

A	B	C	N'
1	0.2	3.7	2.0

260-065

RMR = 9 ln Q' + 44

		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
OVERCUT						
BQFG	A7/R1	19.5	69	1345.5	3.6	70.2
	A5/R2	10.5	68	714	8.3	87.15
TOTAL:		30		2059.5		157.35
				68.7		5.2
UNDERCUT						
BQFG	A7/R1	8	69	552.0	3.6	28.8
	A5/R2	18.5	68	1258.0	8.3	153.6
Pegmatoid	A3/R2	3.5	71	248.5	5.9	20.65
TOTAL:		30		2058.5		203.0
				68.6		6.8

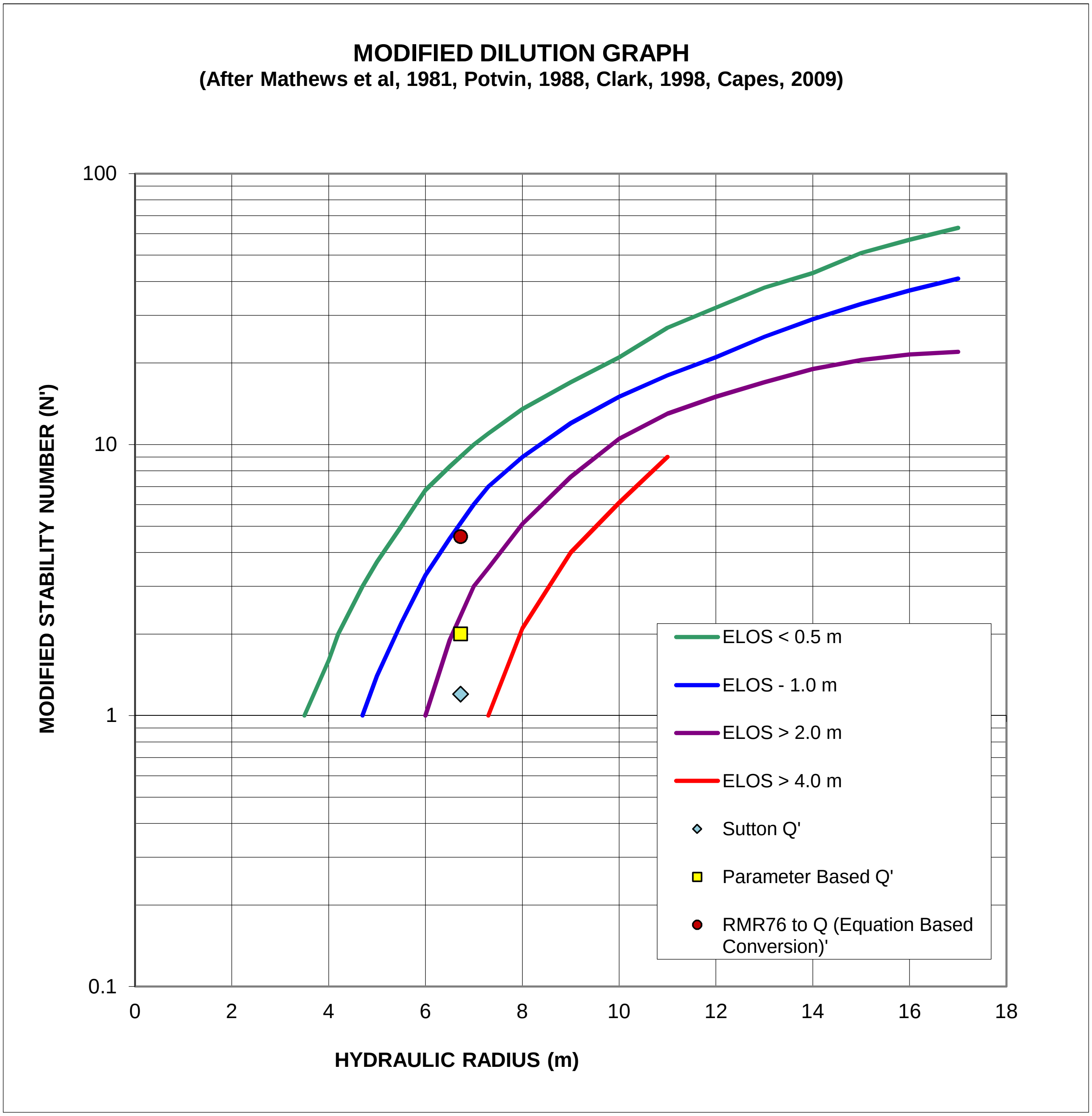
DDH's	RQD in HW:
1723	63
2016	90
2017	70
3309.0	65.0

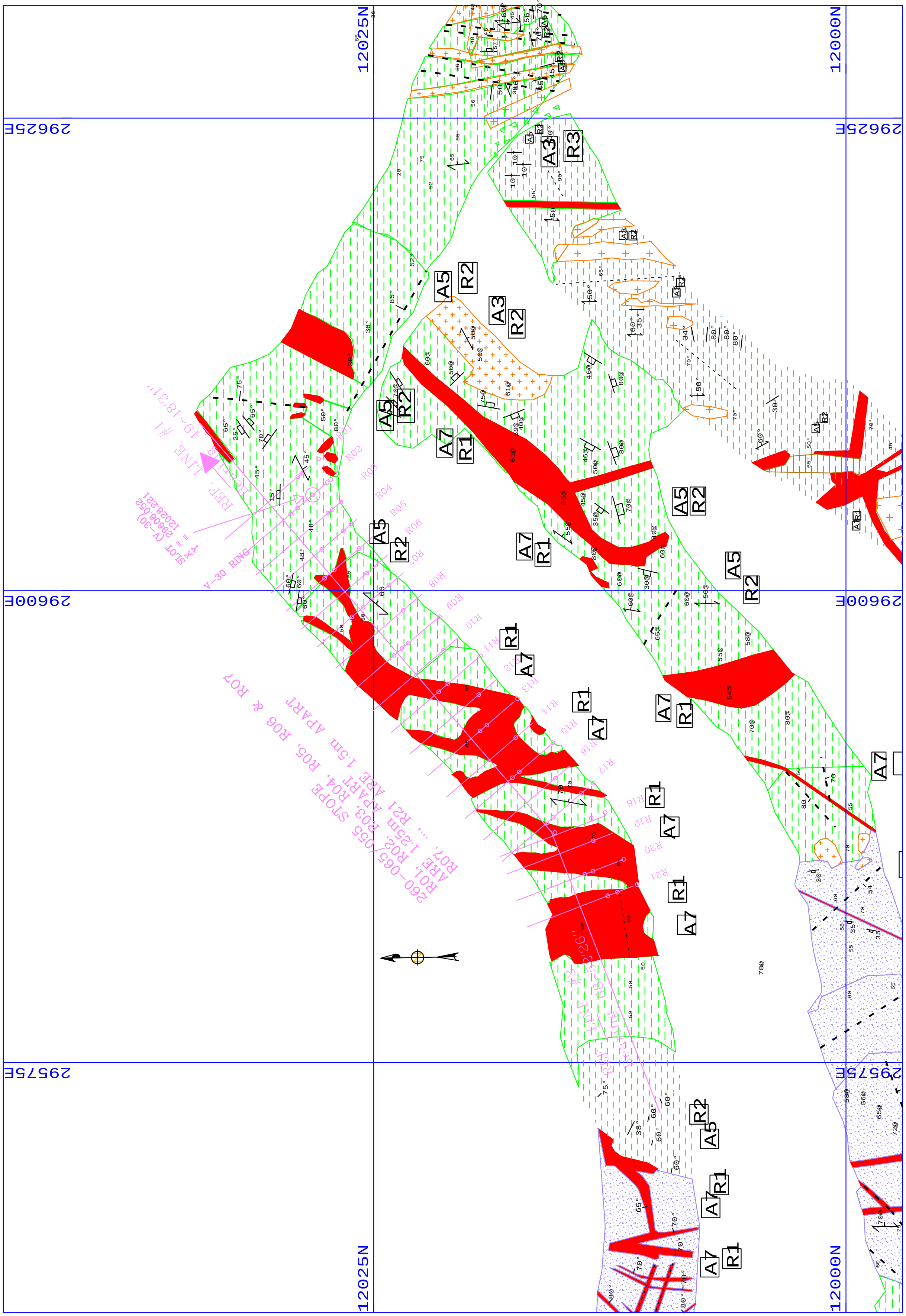
Q' Weighted	RQD	RQD O/C and U/C	Q'
6.2	70.9	68.6	6.0

A	B	C	N'
1	0.2	3.7	4.6

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	260-065	
Modified Stability Number (N')	Sutton N'	1.2
	Parameter Based N'	2.0
	RMR76 to Q (Equation Based Conversion)	4.6
Hydraulic Radius	Actual	6.7
Average HW Overbreak from the Optech:		**1.8** m




12025N
12000N
29625E
29600E
29575E

STOPE	302-2	
Strike Length	21.5	
HW Exposed Height	26.8	
Hydraulic Radius	6.0	
Average HW Dip	41.8	°
Percentage of HW that is:		
Gneissic	11	%
Pegmatoidal	41	%
Other	48	%

302-2 — Sutton's Q' from A/R values

Rock Type	A/R value	Length (m)	RQD	RQD x L	Jn	Jn x L	Jr	Jr x L	Ja	Ja x L	Q' (weighted)
BQFG	A7/R1	4.5	20	90	4	18	0.75	3.375	10	45	
Pegmatite	A7/R1	17	20	340	4	68	0.75	12.75	10	170	
TOTAL:		21.5		430		86		16.125		215	
			RQDw	20.0	Jn(w)	4.0	Jr(w)	0.8	Ja(w)	10.0	0.4

Rock Type	A/R value	Length (m)	RQD	RQD x L	Jn	Jn x L	Jr	Jr x L	Ja	Ja x L	
Calc-Silicate	A7/R1	6	20	120.0	4.0	24.0	0.8	4.5	10.0	60.0	
Calc-Silicate	A5/R2	13.5	75	1012.5	7.5	101.25	1.5	20.25	6	81	
TOTAL:		19.5		1132.5		125.3		24.8		141.0	
			RQDw	58.1	Jn(w)	6.4	Jr(w)	1.3	Ja(w)	7.2	1.6

DDH's	RQD in HW:
2167	77
2245	87
2251	30

Q' Weighted	RQD	Jn	Jr	Ja
1	54	5.2	1.0	8.6

A	B	C	N'
1.0	0.2	3.5	0.9

302-2

Parameter Based Q' from A/R values

Rock Type	A/R value	Length (m)	RQD	RQD x L	Jn	Jn x L	Jr	Jr x L	Ja	Ja x L	Q' (weighted)
BQFG	A7/R1	4.5	75	337.5	9	40.5	0.75	3.375	4	18	
Pegmatite	A7/R1	17	25	425	4	68	0.75	12.75	10	170	
TOTAL:		21.5		762.5		108.5		16.1		188.0	
			RQDw	35.5	Jn(w)	5.0	Jr(w)	0.8	Ja(w)	8.7	0.6

Rock Type	A/R value	Length (m)	RQD	RQD x L	Jn	Jn x L	Jr	Jr x L	Ja	Ja x L	
Calc-Silicate	A7/R1	6	25	150	4	24	0.75	4.5	10	60	
Calc-Silicate	A5/R2	13.5	60	810	7.5	101.25	1.5	20.3	4	54	
TOTAL:		19.5		960.0		125.3		24.8		114.0	
			RQDw	49.2	Jn(w)	6.4	Jr(w)	1.3	Ja(w)	5.8	1.7

DDH's	RQD in HW:
2167	77
2245	87
2251.0	30.0

Q' Weighted	RQD	Jn	Jr	Ja
1.3	55.7	5.7	1.0	7.3

A	B	C	N'
1	0.2	3.5	0.9

302-2

RMR = 9 ln Q' + 44

Rock Type	A/R value	Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
BQFG	A7/R1	4.5	69	310.5	3.6	16.2
Pegmatite	A7/R1	17	38	646	0.3	5.1
TOTAL:		21.5		956.5		21.3
			RQDw	44.5		1.0

Rock Type	A/R value	Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
Calc-Silicate	A7/R1	6	38	228.0	0.3	1.8
Calc-Silicate	A5/R2	13.5	63	850.5	2.5	33.75
TOTAL:		19.5		1078.5		35.6
			RQDw	55.3		1.8

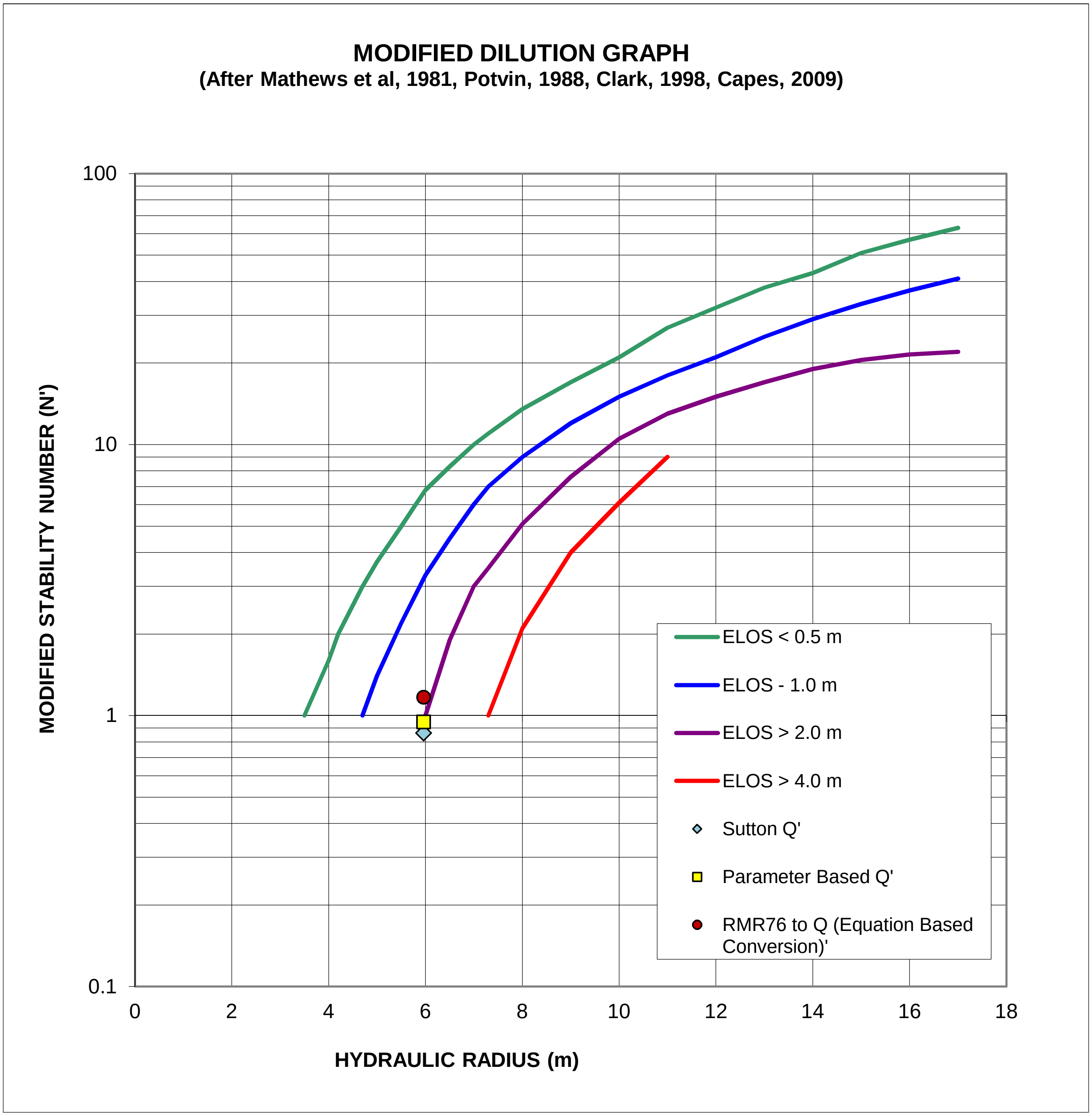
DDH's	RQD in HW:
2167	77
2245	87
2251	30

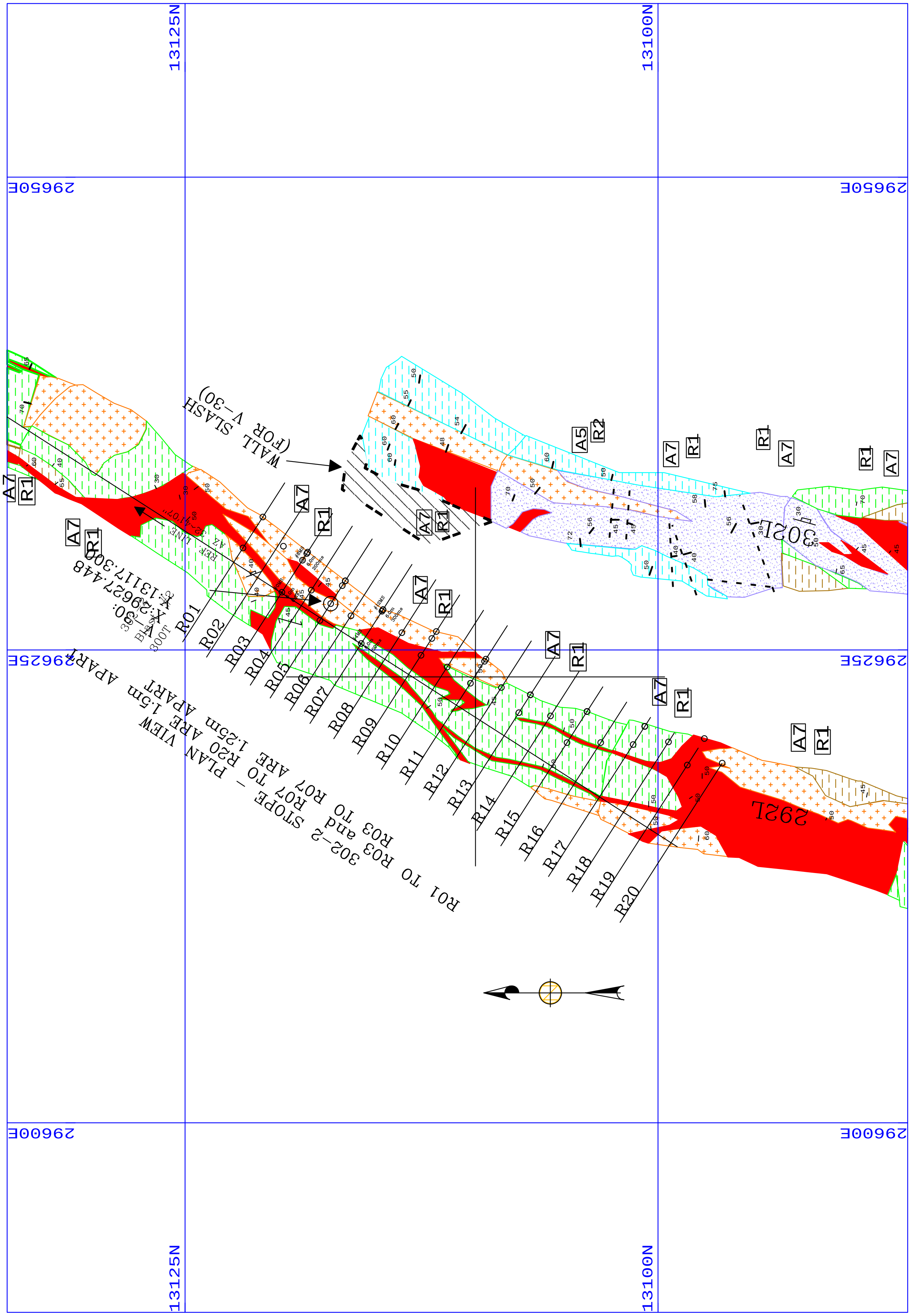
Q' Weighted	RQD	RQD O/C and U/C	Q'
2	59	49.9	1.4

A	B	C	N'
1.0	0.2	3.5	1.2

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	302-2	
Modified Stability Number (N')	Sutton N'	0.9
	Parameter Based N'	0.9
	RMR76 to Q (Equation Based Conversion)	1.2
Hydraulic Radius	Actual	6.0
Average HW Overbreak from the Optech:		**3.2** m




13125N
13100N
29650E
29625E
29600E
302-2 STOPE - PLAN VIEW
R01 TO R03 and R07 TO R20 ARE 1.5m APART
R03 TO R07 ARE 1.25m APART
WALL SLASH
(FOR V-30)
302L
292L
A7
R1
A5
R2
R01
R02
R03
R04
R05
R06
R07
R08
R09
R10
R11
R12
R13
R14
R15
R16
R17
R18
R19
R20

STOPE	362-3	
Strike Length	26	
HW Exposed Height	23.5	
Hydraulic Radius	6.2	
Average HW Dip	31.6	°
Percentage of HW that is:		
Gneissic	3	%
Pegmatoidal	14	%
Other	83	%

362-3

Sutton's Q' from A/R values

OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
PEGMATOID	A7/R1	7.2	20	144	4	28.8	0.75	5.4	10	72	
CALC-SIL	A7/R1	15.7	20	314	4	62.8	0.75	11.775	10	157	
BQFG	A7/R1	1.4	20	28	4	5.6	0.75	1.05	10	14	
QUARTZITE	A7/R1	1.7	20	34.0	4	6.8	0.8	1.3	10.0	17.0	
TOTAL:		26		520.0		104.0		19.5		260.0	
				20.0		4.0		0.8		10.0	0.4
UNDERCUT											
CALC-SIL	A5/R2	8.5	75	637.5	7.5	63.75	1.5	12.75	6	51	
	A7/R1	15.5	20	310.0	4	62.0	0.8	11.6	10.0	155.0	
	A7/R2	2	60	120.0	6.0	12.0	1.5	3.0	6.0	12.0	
TOTAL:		26		1067.5		137.8		27.4		218.0	
				41.1		5.3		1.1		8.4	1.0
									AVERAGE:		0.7

DDH's	RQD in HW:
2719.0	20.0
2246	59
EPE-042	47

Q' Weighted	RQD	Jn	Jr	Ja
0.8	37.4	4.6	0.9	9.2

A	B	C	N'
1.0	0.2	2.6	0.4

362-3

Parameter Based Q' from A/R values

OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q' (weighted)
PEGMATOID	A7/R1	7.2	25	180	4	28.8	0.75	5.4	10	72	
CALC-SIL	A7/R1	15.7	25	392.5	4	62.8	0.75	11.775	10	157	
BQFG	A7/R1	1.4	75	105.0	9.0	12.6	0.75	1.1	4	5.6	
QUARTZITE	A7/R1	1.7	25	42.5	4.0	6.8	0.75	1.3	10.0	17.0	
TOTAL:		26		720		111.0		19.5		251.6	
				27.7		4.3		0.8		9.7	0.5
UNDERCUT											
CALC-SIL	A5/R2	8.5	60	510	7.5	63.75	1.5	12.8	4	34	
	A7/R1	15.5	25	387.5	4.0	62.0	0.8	11.6	10.0	155.0	
	A7/R2	2	50	100.0	6.0	12.0	1.5	3.0	6.0	12.0	
TOTAL:		26		997.5		137.8		27.4		201.0	
				38.4		5.3		1.1		7.7	1.0
									AVERAGE:		0.7

DDH's	RQD in HW:
2719	20
2246	59
EPE-042	47.0

Q' Weighted	RQD	Jn	Jr	Ja
0.8	38.4	4.8	0.9	8.7

A	B	C	N'
1	0.2	2.6	0.4

362-3

RMR = 9 ln Q' + 44

OVERCUT		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
PEGMATOID	A7/R1	7.2	38	273.6	0.3	2.16
CALC-SIL	A7/R1	15.7	38	596.6	0.3	4.71
BQFG	A7/R1	1.4	69	96.6	3.6	5.04
QUARTZITE	A7/R1	1.7	38	64.6	0.3	0.5
TOTAL:		26		1031.4		12.4
				39.7		0.5
UNDERCUT						
CALC-SIL	A5/R2	8.5	63	535.5	2.5	21.25
	A7/R1	15.5	38	589.0	0.3	4.7
	A7/R2	2	73	146.0	4.7	9.4
TOTAL:		26		1270.5		35.3
				48.9		1.4

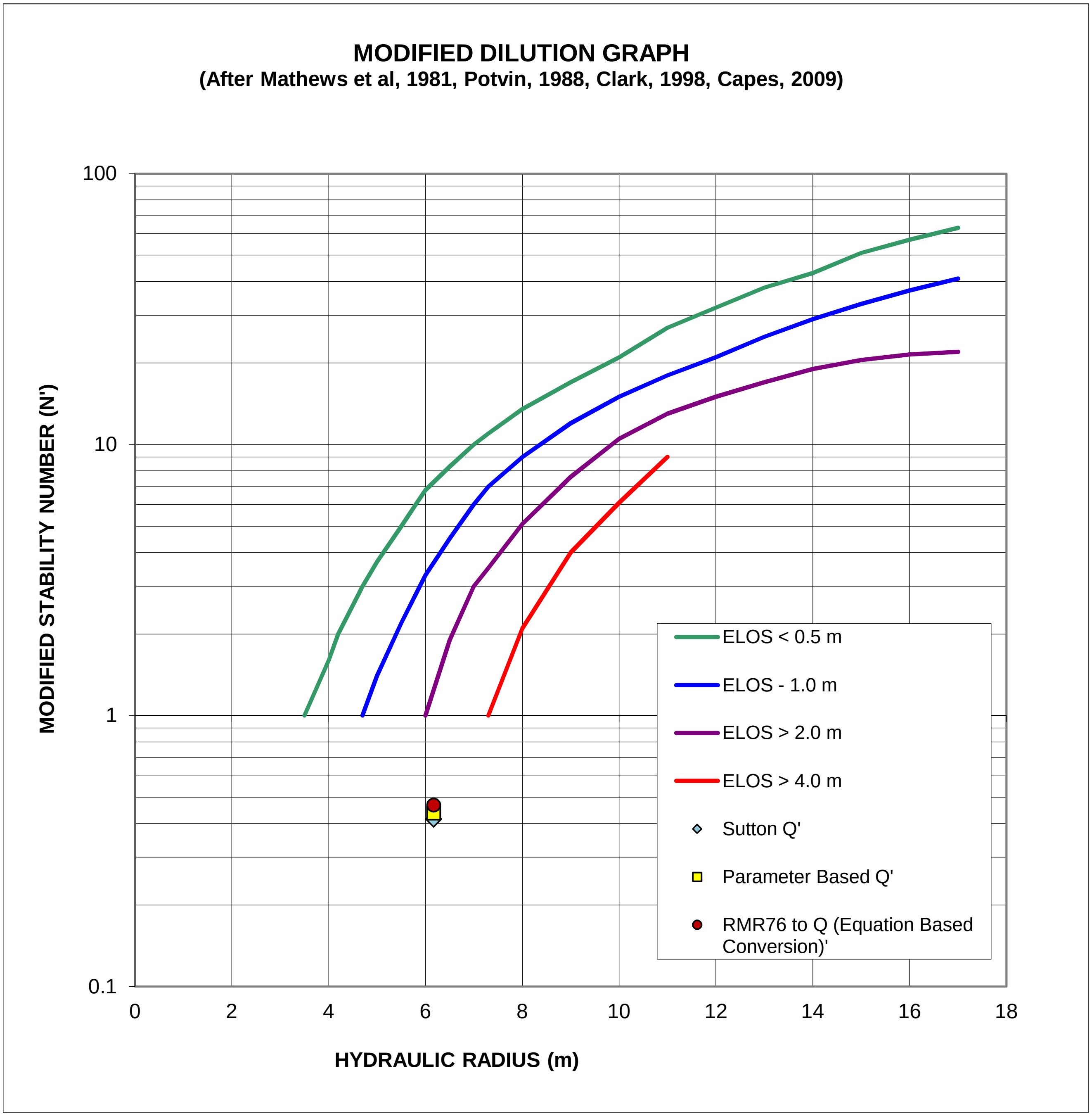
DDH's	RQD in HW:
2719.0	20.0
2246	59
EPE-042	47

Q' Weighted	RQD	RQD O/C and U/C	Q'
0.9	42.9	44.3	0.9

A	B	C	N'
1.0	0.2	2.6	0.5

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	362-3	
Modified Stability Number (N')	Sutton N'	0.4
	Parameter Based N'	0.4
	RMR76 to Q (Equation Based Conversion)	0.5
Hydraulic Radius	Actual	6.2
Average HW Overbreak from the Optech:		**4.3** m



STOPE	362-4	
Strike Length	20.5	
HW Exposed Height	16.9	
Hydraulic Radius	4.6	
Average HW Dip	56.8	°
Percentage of HW that is:		
Gneissic	2	%
Pegmatoidal	98	%
Other	0	%

362-4			Sutton's Q' from A/R values								
OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
Pegmatite	A5/R2	4	75	300	7.5	30	1.5	6	6	24	
	A7/R2	16.5	60	990	6	99	1.5	24.75	6	99	
TOTAL:		20.5		1290		129		30.75		123	
				62.9		6.3		1.5		6.0	2.5
UNDERCUT											
AGGN	A7/R1	1	20	20.0	4.0	4.0	0.8	0.8	10.0	10.0	
Pegmatite	A7/R1	19.5	20	390.0	4.0	78.0	0.8	14.6	10.0	195.0	
TOTAL:		20.5		410		82		15.375		205	
				20.0		4.0		0.8		10.0	0.4

DDH's	RQD in HW:
2708	55

Q' Weighted	RQD	Jn	Jr	Ja
1.3	46.0	5.1	1.1	8.0

A	B	C	N'
1	0	4.7	1.2

362-4

Parameter Based Q' from A/R values

OVERCUT		Length (m)	RQD	RQD (weighted)	Jn	Jn (weighted)	Jr	Jr (weighted)	Ja	Ja (weighted)	Q'
Pegmatite	A5/R2	4	60	240	7.5	30	1.5	6	4	16	
	A7/R2	16.5	50	825	6	99	1.5	24.75	6	99	
TOTAL:		20.5		1065.0		129.0		30.8		115.0	
				52.0		6.3		1.5		5.6	2.2
UNDERCUT											
AGGN	A7/R1	1	75	75.0	9.0	9.0	0.8	0.8	4.0	4.0	
Pegmatite	A7/R1	19.5	25	487.5	4	78	0.75	14.625	10	195	
TOTAL:		20.5		562.5		87		15.4		199	
				27.4		4.2		0.8		9.7	0.5

DDH's	RQD in HW:
2708	55

Q' Weighted	RQD	Jn	Jr	Ja
1.2	44.8	5.3	1.1	7.7

A	B	C	N'
1.0	0.2	4.7	1.2

362-4

RMR = 9 ln Q' + 44

OVERCUT		Length (m)	RQD	RQD (weighted)	Q'	Q' (weighted)
Pegmatite	A5/R2	4	63	252	2.5	10
	A7/R2	16.5	73	1204.5	4.7	77.55
TOTAL:		20.5		1456.5		87.55
				71.0		4.3
UNDERCUT						
AGGN	A7/R1	1	69	69.0	3.6	3.6
Pegmatite	A7/R1	19.5	38	741.0	0.3	5.9
TOTAL:		20.5		810		9.45
				39.5		0.5

DDH's	RQD in HW:
2708	55

Q' Weighted	RQD	RQD O/C and U/C	Q'
2.4	55.2	55.3	2.4

A	B	C	N'
1	0	4.7	2.2

STOPE RECONCILIATION - DILUTION GRAPH

Stope:	362-4	
Modified Stability Number (N')	Sutton N'	1.2
	Parameter Based N'	1.2
	RMR76 to Q (Equation Based Conversion)	2.2
Hydraulic Radius	Actual	4.6
Average HW Overbreak from the Optech:		**0.4** m